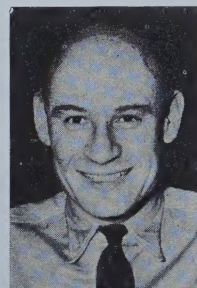


**L. W. Eastwood** (p. 638); born in Wiota, Wis., and went to Palloville high school. Graduated from University of Wisconsin with B.S., M.S., Ph.D. degrees. Is a member of AIME and lives in Westerville, Ohio. Became Asst. Professor, Michigan College of Mining & Technology; Research Metallurgist of Aluminum Co. of America, and at present is Supervisor Magnesium Foundry, Batelle Memorial Institute. Mr. Eastwood previously has presented several AIME papers. . . . **George D. Sells** (p. 646); born in Bellview, Pa., attended Ben Avon High School as well as the Washington and Jefferson College. He received his B.S. degree from Pennsylvania State College. His career includes service with the American Steel & Wire Co. in Cleveland and Duluth. He is now with the Pittsburgh Steel Co. at Monessen, Pa. as superintendent, and lives at Belle Vernon. Is an AIME member and likes to play golf and putter in his garden. . . . **B. D. Cullity** (p. 648); born in Havre, Montana, attended Loyola High School in Montreal; McGill University, University of Minnesota and MIT, receiving degrees of B.Eng., M.S., and Sc.D. He has been an instructor in Metallurgy at the Montana School of Mines and the University of Minnesota; group leader on the Manhattan Project at MIT, research metallurgist at Ecole des Mines, Paris, and scientific liaison officer for the Office of Naval Research in London. He now lives in South Bend, Ind., and is an AIME member. He has presented several other papers before the AIME. His hobby is painting. . . . **Kurt Neustaetter** (p. 651); born in Stuttgart, Germany, and attended high school in Stuttgart and college in Berlin, where he received a degree of D. Eng. in 1932. After serving a short apprenticeship in Germany he joined the Inland Steel Co. at East Chicago, Ind., beginning as blast furnace apprentice, and is now blast furnace engineer. He resides in Chicago and is an AIME member. Received the AIME J. E. Johnson, Jr. Award in 1947. Likes to collect stamps and climb mountains. . . . **R. X. Touzalin** (p. 656); born in Chicago, attended high school in Joliet and the Lyons Township Junior College at La Grange. He received his S.B. from MIT and worked on Carrie Furnaces at Carnegie-Illinois Steel Corp., as practice apprentice and stock house foreman. He is now service engineer for A. G. McKee & Co. He is not an AIME member. Likes to play golf, to fish, and is a model railroader. . . . **E. McL. Tittman** (p. 669); born in St. Paul, Minn., attended high school in Cornwall, N. Y. He received a B.S. degree at MIT. His entire career has been with the American Smelting & Refining Co.—at Salt Lake City, East Helena, Montana, and El Paso, Texas—from chemist to manager. He now lives in El Paso. Is an AIME member and likes to swim, play golf, fish and hunt. . . . **R. W. Gurry** (p. 671); born in Schenectady, N. Y., where he attended high school. He received his B.S. in chemistry from Union College and his Ph.D. in Physical Chemistry from Yale University. Since 1937 he has worked in the Research Laboratory of the U. S. Steel Corp., at Kearny, N. J. He lives in Westfield, N. J. Not an AIME member but has presented other papers before the AIME, including: "The Solubility of Carbon as Graphite in Gamma Iron" and "The De-



B. D. Cullity



K. Neustaetter



R. W. Gurry





Published to provide a continuing, authoritative, and up-to-date record of technologic, engineering, and economic progress in all branches of the metals industry by the

Metals Branch

American Institute of Mining and Metallurgical Engineers, Inc.  
29 West 39th Street, New York 18

○ ○ ○

**T. W. LIPPETT**

Manager of Publications  
Editor, Journal of Metals

○ ○ ○

**JAMES A. STANGARONE**

Advertising Manager

○ ○ ○

**WINIFRED D. GIFFORD, HAROLD N. UNDERHILL, Assistant Editors; GRACE PUGSLEY, Production Manager; WALTER J. SEWING, Assistant.**

○ ○ ○

#### OFFICERS, AIME

D. H. McLaughlin	President
A. B. Kinzel	Vice President
Philip Kraft	Vice President
R. W. Thomas	Vice President
C. H. Benedict	Vice President
Andrew Fletcher	VP & Treasurer

○ ○ ○

#### AIME STAFF

E. H. Robie	Secretary
H. A. Maloney	Asst. Treasurer
E. J. Kennedy, Jr.	Asst. Secretary
Ernest Kirkendall	Asst. Secretary
H. Newell Appleton	Asst. to Secretary

○ ○ ○

#### Technical Publications Committee

F. B. Foley	Chairman
E. J. Kennedy, Jr.	Secretary

○ ○ ○

#### Auxiliary Publications Committee

O. B. J. Fraser	Chairman, IMD
Michael Tenenbaum	Chairman, ISD
Philip T. Stroup	Chairman, EMD

○ ○ ○

#### Publications Advisory Committee

J. B. Austin, F. N. Rhines, C. D. King,  
John D. Sullivan, Fred P. Peters, A. W.  
Schlechten, Frank T. Sisco, M. Tenenbaum,  
Felix Wormser

○ ○ ○

The AIME Also Publishes  
Mining Engineering  
and

Journal of Petroleum Technology

# METALS

## IN THIS ISSUE

Vol. 188, NO. 4

APRIL, 1950

### COVER

A new type of precision 4-high cold rolling mill. The work rolls are small and the back-up rolls are driven (not the work rolls, as is the usual procedure) which permits pouring in a great quantity of power to secure heavy reductions. This high-speed mill shows particular promise for rolling silicon, high-carbon and stainless steels to intermediate and light gages. Mill design and construction are by Cold Metal Products Co., Youngstown.

### FEATURES

Authors in This Issue	620
Engineering Employment Service	624
Journal of Metals REPORTER	627
It's Everyone's Business	628
Drift of Things	630

### NEWS

Board of Directors	628B	Personals, Obituaries	632
AIME Section Activities	628D	New Members	632

### TECHNICAL ARTICLES

Continuous Method of Producing Ductile Titanium		634
<i>P. J. Maddex, L. W. Eastwood</i>		
Use of Adirondack Sinter in Blast Furnaces	<i>E. H. Riddle</i>	641
Blowing Out of a Carbon-Hearth Blast Furnace	<i>G. D. Sells</i>	646
Blowing Out Carbon-Hearth Furnaces	<i>H. P. Saxer</i>	647
European Research in Physical Metallurgy	<i>B. D. Cullity</i>	648
Conditions of Outstanding Blast Furnace Operation	<i>Kurt Neustaetter</i>	651
Application of Superimposed Precipitators	<i>R. E. Touzalin</i>	656

### TRANSACTIONS

Extractive Metallurgy of Aluminum	R. S. Sherwin	661
Water Sealed Wind Boxes for Dwight and Lloyd Sintering Machine	E. McL. Tittman, E. A. Hase	669
Composition of Atmospheres Inert to Heated Carbon Steel	R. W. Curry	671
Thermal Expansion Characteristics of Stainless Steels	Between $-300^{\circ}$ and $1000^{\circ}\text{F}$ D. E. Furman	688
Acid Conditioning of Metallurgical Smoke for Cottrell Precipitation	A. L. Labbe	692
Experimental Operation of a Basic-Lined Surface-Blown Hearth for Steel Production	C. E. Sims, F. L. Toy	694

#### Technical Note

Note on Surface Diffusion in Sintering of Metallic Particles	N. Cabrera	667
--	------------	-----

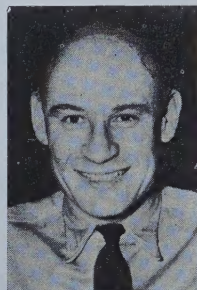
Published the first day of each month by the American Institute of Mining and Metallurgical Engineers, Inc., 29 West 39th Street, New York 18, N. Y. Telephone: Pennsylvania 6-9220. Subscription \$8 per year for non-AIME members in United States and North, South and Central America; \$9, foreign; \$6 for AIME members, or \$4 in combination with a subscription to *Mining Engineering* or the *Journal of Petroleum Technology*. Single copies, 75 cents; special issues, \$1.50 . . . The AIME is not responsible for any statement made or opinion expressed in its publications. . . . Copyright 1949 by the American Institute of Mining and Metallurgical Engineers, Inc. . . . Registered cable address, AIME New York . . . Indexed in Engineering Index, Industrial Arts Index and Chemical Abstracts . . . Reentered as second-class matter Sept. 21, 1949, at the post office at New York, N. Y., under the Act of March 3, 1879.



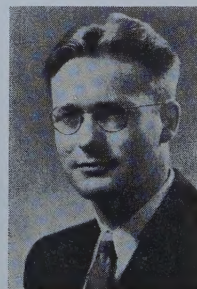
**L. W. Eastwood** (p. 638); born in Wiota, Wis., and went to Palloville high school. Graduated from University of Wisconsin with B.S., M.S., Ph.D degrees. Is a member of AIME and lives in Westerville, Ohio. Became Asst. Professor, Michigan College of Mining & Technology; Research Metallurgist of Aluminum Co. of America, and at present is Supervisor Magnesium Foundry, Batelle Memorial Institute. Mr. Eastwood previously has presented several AIME papers. . . . **George D. Sells** (p. 646); born in Bellview, Pa., attended Ben Avon High School as well as the Washington and Jefferson College. He received his B.S. degree from Pennsylvania State College. His career includes service with the American Steel & Wire Co. in Cleveland and Duluth. He is now with the Pittsburgh Steel Co. at Monessen, Pa. as superintendent, and lives at Belle Vernon. Is an AIME member and likes to play golf and putter in his garden. . . . **B. D. Cullity** (p. 648); born in Havre, Montana, attended Loyola High School in Montreal; McGill University, University of Minnesota and MIT, receiving degrees of B.Eng., M.S., and Sc.D. He has been an instructor in Metallurgy at the Montana School of Mines and the University of Minnesota; group leader on the Manhattan Project at MIT, research metallurgist at Ecole des Mines, Paris, and scientific liaison officer for the Office of Naval Research in London. He now lives in South Bend, Ind., and is an AIME member. He has presented several other papers before the AIME. His hobby is painting. . . . **Kurt Neustaetter** (p. 651); born in Stuttgart, Germany, and attended high school in Stuttgart and college in Berlin, where he received a degree of D. Eng. in 1932. After serving a short apprenticeship in Germany he joined the Inland Steel Co. at East Chicago, Ind., beginning as blast furnace apprentice, and is now blast furnace engineer. He resides in Chicago and is an AIME member. Received the AIME J. E. Johnson, Jr. Award in 1947. Likes to collect stamps and climb mountains. . . . **R. X. Touzalin** (p. 656); born in Chicago, attended high school in Joliet and the Lyons Township Junior College at La Grange. He received his S.B. from MIT and worked on Carrie Furnaces at Carnegie-Illinois Steel Corp., as practice apprentice and stock house foreman. He is now service engineer for A. G. McKee & Co. He is not an AIME member. Likes to play golf, to fish, and is a model railroader. . . . **E. McL. Tittman** (p. 669); born in St. Paul, Minn., attended high school in Cornwall, N. Y. He received a B.S. degree at MIT. His entire career has been with the American Smelting & Refining Co. — at Salt Lake City, East Helena, Montana, and El Paso, Texas — from chemist to manager. He now lives in El Paso. Is an AIME member and likes to swim, play golf, fish and hunt. . . . **R. W. Gurry** (p. 671); born in Schenectady, N. Y., where he attended high school. He received his B.S. in chemistry from Union College and his Ph.D. in Physical Chemistry from Yale University. Since 1937 he has worked in the Research Laboratory of the U. S. Steel Corp., at Kearny, N. J. He lives in Westfield, N. J. Not an AIME member but has presented other papers before the AIME, including: "The Solubility of Carbon as Graphite in Gamma Iron" and "The De-



B. D. Cullity



K. Neustaetter



R. W. Gurry



oxidizing Power of Boron in Liquid Steel." He enjoys music, bowling and sailing. . . . **C. E. Sims** (p. 694); born in Chicago where he attended Crane Technical High School. He received his B.S. in Chemical Engineering from the University of Illinois and an M.S. in Metallurgy from the University of Utah. His career includes service in the U. S. Bureau of Mines as Associate Metallurgist; Assistant Director of Research in the American Steel Foundries, and Assistant Director of Batelle Memorial Institute in Columbus, where he now resides. He is an AIME member and has presented many previous papers before AIME. He enjoys fishing and golfing. . . . **F. L. Toy** (p. 694); born in Pittsburgh where he attended high school and East Liberty Academy. He received degrees in Electrical and Metallurgical Engineering from Lehigh University. His entire career has been with the Carnegie-Illinois Steel Corp. where he began as foreman of an electric furnace and is now assistant to the manager of research and development. His home is in Edgewood, Pa. Is an AIME member and likes to fish, watch birds and putter in his garden. . . . **P. J. Maddex** (p. 634); born in Mechanicsburg, Ohio, attended high school there, and received Bachelor of Chemical Engineering degree. Was Radio Tube Designer for Ken-Rad Tube and Lamp Corp., Owensboro, Kentucky; Chemical Engineer with General Electric Company, Bowling Green, Kentucky; Research Engineer at Battelle Memorial Institute, and presently development engineer, National Lead Company, Titanium Division, South Amboy, New Jersey. Mr. Maddex lives in Fords, New Jersey. His chief diversion is golf and he has a penchant for gardening. . . . **D. E. Furman** (p. 688); born in Annadale, Staten Island, New York, attended Brooklyn Technical High School, received B. ME degree from New York University. He lives on Staten Island and is on the staff of the research laboratory of the International Nickel Company, Bayonne, New Jersey. Mr. Furman's chief diversion is fishing.



F. L. Toy

## ATTENTION—IMD AUTHORS

The Fall Meeting of the Institute of Metals Div., AIME, will be at the Sheraton Hotel, Chicago, Oct. 23-25. Deadline for papers for this meeting is April 15, 1950. Because of the large number of papers already published, the publication funds available to the I.M.D. for the balance of this year will not be sufficient to publish the usual number of fall meeting papers. A limited number of papers can be printed prior to the fall meeting. Some fall meeting papers may be accepted for publication but not printed this year. The date of receipt will determine the order of publication, space permitting. A number of special sessions are being planned, which should make the fall meeting more interesting than those of the past.

\* \* \*

The situation with respect to funds available for publishing IMD papers is now somewhat brighter than it was a month ago. There are good prospects of finding money to publish this year all of the acceptable papers that may be submitted for the Fall Meeting, if economies are practiced. It will be necessary to ask authors to condense their papers as much as possible. It is thought that in almost all instances this will result in improvement, with no sacrifice of quality or suppression of experimental results. As to the future, there is no reason to doubt that the AIME will be able to publish promptly all the good papers that are submitted, by continuing this policy of reasonable economy of publication space. The deadline for submission of Fall Meeting papers for the IMD is April 15, 1950.



- \* The United States has authorized the Japanese to set up trade missions in New York, San Francisco, Los Angeles and Honolulu. This is the first step along the road for Japan to regain diplomatic recognition abroad. Japan has been permitted also by occupation authorities to take part in international technical conferences.
- \* In transporting radioactive isotopes, air freight charges to distant countries can be quite high, what with say 30 lb of lead packing around the small quantity of radioactive material. The South African National Physical Laboratory has done away with lead sheathing and the South African Airways carry the radioactive material in containers on the wing tips, some 50 ft away from the passengers, crew and other cargo.
- \* Tests on 1000-lb charges of hot metal showed that surface blowing in a basic-lined hearth to produce steel from basic iron requires no special compositions of hot metal. Mechanical properties of the steel produced are such as to place them in the same limits as openhearth steel of a similar composition. Process time is about 12 min, blast pressures are less than 3 psi and up to 97 pct of the phosphorus and 50 pct of the sulphur can be eliminated. (p. 694)
- \* Water sealed wind boxes for Dwight and Lloyd sintering machines granulate lead, permitting its easy removal, in double roasting of sinter carrying high percentages of lead concentrates. This eliminates the problem of removing sheets of metallic lead formed in the wind boxes. (p. 669)
- \* A newcomer into the field of cold extrusion of steel is Fulton-Sylphon Co., working on Ordnance material including projectiles.
- \* Tests using a high quality sinter mix, composed of blast furnace flue dust and 40 to 50 pct Port Henry Magnetite concentrate, have been made at Jones & Laughlin's Aliquippa No. 5 furnace. Indications are that high blast temperatures can be used when operating on a high percentage of this sinter in the burden. (p. 641)
- \* A continuous method of producing titanium, using magnesium and titanium tetrachloride as raw materials, has evolved from experimental work sponsored by the Air Materiel Command. (p. 634)
- \* The Venezuelan Trade Mission to the United States announced that Bethlehem Steel will begin exporting iron ore from its El Pao concession, south of the Orinoco River, in about three months. The ore will be transported by rail from the mines to barges on the Orinoco and trans-shipped to ore vessels at the coast for shipment to Sparrows Point, Md.
- \* Increased demand for magnesium sheet in aircraft construction and other phases of the national security program have resulted in Alcoa establishing a sheet magnesium rolling division at its New Kensington, Pa., works. Operation will begin shortly.
- \* Novel prospecting methods being used by the USGS in its search for uranium include shock-mounted instruments in light trucks, Geiger counter probes lowered into auger holes, and experimental airborne prospecting.
- \* A compact non-broadcast television system for industrial, scientific and educational use was unveiled by RCA at the Institute of Radion Engineers Convention in N.Y. The system, eventually available in color as well as black and white, features the Vidicon, new pick-up tube only 1 in. diam.



# It's Everyone's Business

**M**ARCH 15—Industry is rapidly snapping back from another coal crisis, other business news is in general favorable and the outlook through the Spring is by most observers considered quite promising. Most industries report comfortable backlogs of unfilled orders, department store sales in recent weeks have equalled in dollars those of the corresponding period last year, retail sales of automobiles show again over the same period a year ago, and new orders for machine tools are at the highest figures since August 1946.

Orders for railway freight cars have shown a very encouraging pickup, with a total of 9385 cars in January and the N. Y. Central alone ordering 4500 in February. Construction continues very active. In January new housing starts totaled 80,000 against 50,000 a year ago and the daily average contract awards, according to the Dodge reports, were 57 pct larger. In the first 22 days of February the increase in awards was 27 pct. Lumber orders since Jan. 1 have run 22 pct above last year.

All of these favorable factors bolster the opinion that the upturn in industrial output and employment which began last summer has not yet spent itself and that the setback caused by the strikes will be made up when people get back to work. Only in a few lines are there indications that primary production may be outrunning the consumption of finished goods, although inventory build-up would seem to be rather modest as the National Association of Purchasing Agents reported in February that 78 pct of its members were buying for 60 days or less.

Industrial prices as a whole show no significant trend despite the continued alarm over both the immediate and the long-term possibility of inflation caused primarily by the government's increasing expenditure and its inability to balance the budget. A recent advance of 1.9 pct in primary market prices is viewed by some people as the surface symptom of renewed inflation, but even so it is more than two years since the all-time high was reached by the daily index of sensitive commodities in November 1947. Over two years have passed since prices received by farmers hit their peak and about a year and a half since all wholesale prices and the consumer price index touched their highest points in August 1948.

The decline in the wholesale price index has been almost continuous, interrupted only twice by slight advances in the monthly figures from February to March 1949, and, later in the year, from August to September. Of course, this continued decline in the last half of 1949 largely reflects a downward movement in farm and food prices. Wholesale prices for commodities other than farm and food leveled out after June and now stand only slightly under the June level. Farm

products are down by 22 pct from January and by 18.6 pct from August 1948. Food prices are down only slightly less. Consequently, the farm price support program is in serious trouble and has imposed a heavy financial burden on the Treasury.

In those commodities other than farm products only building materials, chemicals and textiles have come down as much as the average, while housefurnishings and metals and their products have fallen hardly at all.

Glancing away from the domestic front, the results of the British general election indicated a highly inconvenient situation, that public opinion there is so exactly deadlocked as to divest the victors of much ability to make forceful decisions or create significant policy. It is generally considered that another election in the near future would give substantially the same result. The British people are equally divided and the politicians are not likely to precipitate another appeal to the country until there is manifest reason to believe that external and internal circumstances have changed sufficiently to permit one party or the other to obtain a decisive verdict.

Because of the lack of a clear mandate steel nationalization will likely lag in Britain. In the meantime there is considerable discussion going on as to whether the European steel industries as a whole are at a disadvantage in world competition with the American industry. An article in the "Statistical Bulletin" for January of the British Iron and Steel Federation takes issue on this point with the authors of the Geneva report, observing that they wrote before devaluation, and this particular conclusion "has been overtaken by events." The "Bulletin" also contends that the statement was in any case incorrect so far as Britain was concerned, and that it was based on scanty and insufficient evidence. To support the rebuttal a detailed and extensive comparison is made of the home prices of certain steel products in Britain and in other countries of the world. The accompanying table gives a selection of the prices published. It sets out the changed position since devaluation and since the recent increases in American steel prices. These are home market prices; no doubt in competitive export business

Comparison of Domestic Steel Prices in Various Countries

	Angles*	Plates*	Heavy Bars*	Rails†	Sheets‡
	5" x 5" x 1/2"	20' x 5' x 1/4"	4" dia.	90 lb. per yd. 60" lengths	6' x 3' x 20 SWG
	£ s. d.	£ s. d.	£ s. d.	£ s. d.	£ s. d.
UK .....	19 13 6	21 14 6	21 13 6	19 2 6	28 6 0
USA .....	33 4 0	34 0 0	36 0 0	29 0 0	39 12 0
Germany ....	21 11 1	22 10 6	21 0 9	21 8 6	32 10 7
Belgium ....	23 16 1	25 18 11	23 17 6	25 8 0	...
Canada .....	30 18 2	30 18 2	32 0 0	...	38 3 8
Sweden .....	26 9 3	...	28 0 3	...	33 13 8
Australia ...	18 5 0	19 5 0	18 15 0	17 12 6	37 0 0

\* 5 tons of a size; 20-ton lots. † 500 tons of a size. ‡ 25 tons of a size.  
Source: British Iron & Steel Federation.



some of the price levels could be brought closer together at need.

While Mr. Atlee barely squeezed through in England and King Leopold just made it in Belgium, over in Moscow Mr. Stalin got another clear mandate from his people. The "da" vote was overwhelming, a result which apparently was rather generally anticipated by major Russian newspapers.

A week or so prior to his trial by vote, Mr. Stalin bade a fervent goodbye to China's Mao Tse-tung after surprisingly long drawn-out negotiations on a trade treaty. Behind the published clauses of the treaty, there lies an attempt by Russia and the Stalinist Chinese leaders to change China's economy, and to exploit the strategically and industrial important areas of Manchuria and Sinkiang for the benefit of the Soviet Union. It is obviously Moscow's intention to make the strategic and economic development of Manchuria fit its own purposes. In Mongolia, but even more in Siberia, the Russians are developing a military position for which an industrialized Manchuria will be the main supply base.

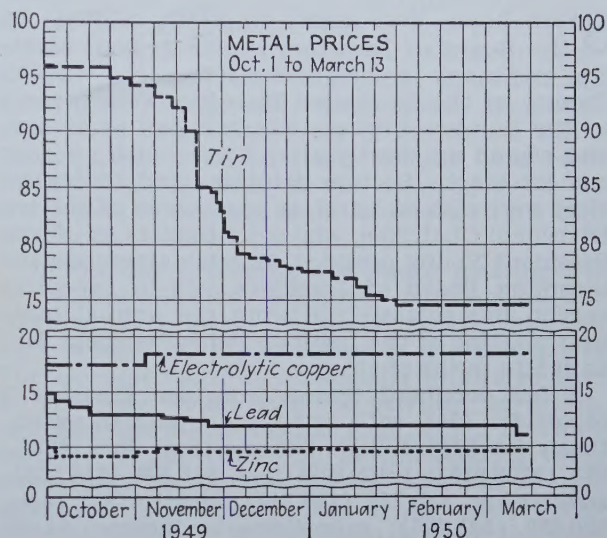
Already the links are close; estimates based on many reports from the spot put at 300,000 the number of Russian officials, and industrial and agricultural experts, who have displaced the half-a-million Japanese that were there in 1945. Most of Manchuria's trade is with the Soviet Union; its wheat, soya beans, meat and milk are exchanged for Soviet clothing, lumber and fuel oil. Private trade has shrunk almost to vanishing point as the Communist "Trade Trust" has expanded its operations. The plan is quite simple. Siberia milks Manchuria and Manchuria will milk China.

And, while Mao Tse-tung tries to explain all this to the Chinese, one of Stalin's own bully-boys has had to do some explaining on orthodox socialistic doctrine in a manner strange to outsiders. Mr. Andreyev, a member of the Politburo since 1931 holding a brief for agriculture, has been tartly criticized in "Pravda." He thought things over for ten days and in a letter to the editor promised to reverse the policy he has enforced for ten years and get along a little faster on farm collectives. This is an excellent example of how Stalin checks on his subordinates. Although the Politburo is so small and although Andreyev is intrusted with responsibility for agriculture, somebody has apparently been detailed to keep watch on him and his policies and to reprove him when necessary in "Pravda."

In Berlin the Russians are tightening down the screws again. Western German businessmen who have in the past few months concluded barter deals with firms in Eastern Germany have been told "off the record" that they must undertake not to let Western Berlin industries have any of the goods or raw materials delivered by the Eastern German firms. This Russian maneuver has as its object the killing-off of Western Berlin's economy. Western Berlin is living on continued injections of ERP and D-marks from the Federal budget (to the extent of \$60 million a year and D-marks one billion), which the Russians doubtless envisage will stop by 1952 at the latest. Then the unemployed of the Western sectors of Berlin will, it is hoped in Moscow, be ready to submit to

the Communists. There is not a doubt that the Communist German spokesmen were uttering a serious threat when they declared recently that the "democrats" of the city must carry out as an urgent task the eliminating of the "imperialist bridgehead" in Berlin.

And, down in France, as if they don't have bigger things to worry about, the Council of the Republic has had before it and rejected a Bill, passed by the National Assembly with a show of hands, that would give the Ministry of Health unlimited discretionary powers over any bever-



age "derived in whole or in part from plants or vegetable extracts." Coca-Cola is not mentioned by name, and the Minister would have to consult the Council of Hygiene and the National Academy of Medicine, but everyone knows what is intended.

According to the London Economist, it is not easy to separate the streams of interest and indignation that have poured into the resistance movement against "coca-colonialism." The wine-growers in Hérault appear to have made the first protests in defense of wine; presumably it was not they who argued that Coca-Cola is injurious to health because they themselves produce local wines whose effect on the liver cannot be entirely in the national interest. Then the Communists took up the defense of national health and the attack against "American imperialism" and soon had the press in full cry. Then the Prime Minister's own Progressive Republicans got involved and a Bill was sponsored by their M. Boulet from Montpellier. But M. Boulet soon found himself attacked from the flank by the Communists, who moved an amendment to his Bill naming Coca-Cola and banning it under the terms of the law. This, of course, had to be opposed, because the measure would then have constituted discrimination against an American product, which is ruled out by existing trade agreements.

So a new and not altogether minor action has begun in the cold war. The French having let off steam, and the Americans having had immense free publicity, they should call it a day. For any law that is passed can be evaded—for example, by adding rum—and any reprisals by the Americans will delight their enemies in France.





## Board Discusses Variety of Subjects at Annual Meeting

Seven hours were spent on AIME business by the Board of Directors at the Annual Meeting, and many important matters were reviewed. Twenty of the 26 elected Directors were present at the Sunday meeting, which began at 10 a.m. and wound up shortly after 5 p.m., with an hour out for lunch. Section delegates and Directors-elect were also on hand, as were seven of the ten Divisional Chairmen, who are Directors ex officio. President Young presided. Monday afternoon the incoming Board of Directors met in executive session immediately following the annual business meeting of the Institute, with President McLaughlin in the chair.

A few pertinent figures from the Treasurer's report for 1949, with comparable data in parenthesis for 1948, follow: Income from dues, \$225,973 (\$208,015); initiation fees, \$16,888 (\$14,562); advertising, \$85,721 (\$68,344); publication sales, \$20,889 (\$28,727); miscellaneous income, \$4,069 (\$9,064); total income, \$393,133 (\$328,711). Expenses for publications, \$300,364 (\$236,916); Institute expenses (rent, library, Local Sections, Divisions, Branches, meetings, Secretary's traveling, insurance, pension) \$83,137 (\$104,871); General Office expense (salaries not otherwise allocated, accounting, membership, telephone, stationery, postage, furniture, miscellaneous) \$90,450 (\$80,157); total expense, \$473,951 (\$424,862). The income figures do not include \$8,769 (\$11,493) in the form of contributions. The operating deficit for 1949 was thus \$72,048, compared with \$84,658 in 1948. Treasurer Fletcher pointed out that the large increase in publications expense was an investment in better magazines for AIME members, which in future years should bring in more advertising revenue. The preliminary budget for 1950, which Mr. Fletcher also offered, indicates a balance in income and expenses because of a substantial increase in income expected because of the rise in dues and a larger volume of advertising. This can be more accurately established when the final budget is adopted at the April meeting of the Board. The Petroleum Branch, in particular, outlined a plan whereby it might be possible to reduce printing costs and increase advertising income in 1950.

A Controller for the Institute, in the person of George I. Brigden, without salary, as recommended by the Executive and Finance Committees, was appointed to review Institute accounting procedures.

A report of the ECPD Committee on Professional Recognition, recommending that all of the Founder Societies adopt uniform grades of membership, was discussed, but in view of the work involved in reclassifying all present Institute

members it was voted not to adopt the proposal.

Considerable discussion arose from a proposal from the Mineral Economics Division, offered by Elmer Pehrson, that the AIME policy on controversial matters, as printed on page 85 of the 1948 Directory, be considerably broadened. Various arguments for and against were offered. The matter was considered so important that action was postponed until the April meeting of the Board to give time for further study.

At a previous Board meeting it had been voted that all Institute, Divisional, and Branch meetings be self-supporting so far as the AIME treasury was concerned, the idea being that no part of a member's dues should be used to finance a meeting which he might not be able to attend. Definite rules for financing and apportioning the cost of such meetings were adopted by the Board at this meeting and copies will be sent to all those concerned.

The plight of young men who had been in World War II and had subsequently completed their education, but who could not become Junior Members because they had passed the 30-year age limit, was discussed by Mr. Head. He advocated granting them permission to be admitted as Junior Members for a period up to three years following granting of their Bachelor's degree, provided only they had not reached the age of 33. Notice was given of an appropriate change in the bylaws to permit this action.

Edward L. Clark, State Geologist for Missouri, presented the essentials of a project now under way by the Engineers Joint Council Committee on National Water Policy, in which AIME participation was desired. The aim of the committee is to give engineering advice to a Presidential Commission that is studying the question. Some difference of opinion was expressed as to the advisability of participating in the work of this committee. Further, it was estimated that the cost to the AIME would be several hundred dollars. It was finally voted to defer action.

A desire to make the Board of Directors more truly representative of all geographical and professional interest groups, and to improve its functional efficiency, led to the appointment of a committee consisting of W. M. Peirce, chairman, E. R. Price and Lloyd E. Elkins, which reported at this meeting. It recommended that for the time being the number of elected Directors continue at 27, with the Divisional Chairmen Directors ex officio. It further recommended that the Nominating Committee be reduced from 19 to 18. Three of these would be appointed by the President, as in the past. Eleven of the remainder would be chosen by the Council of Section



Delegates. Of the other four, two would be chosen by the Mining Branch, and one each by the Metals and Petroleum Branches. Heretofore, the Local Sections have named all members of the Nominating Committee except the three selected by the President.

Of the 27 elected Directors, the Committee felt that the President and past Presidents should not be considered as representing any geographical district, leaving 24. Further, it was believed desirable that the Nominating Committee each year have three Directorships (for a three-year term) at its disposal which are not preempted by fixed apportionment. This leaves fifteen. Of this number, it felt that six should be residents of the New York metropolitan district in order that they can be readily available for the day-to-day work of the Institute, and so that a good representation of Directors may be had at New York Board meetings. Of the remaining nine, it was recommended that one should come from each of nine other geographical districts, which have been delimited to secure an approximate equal number of members in each. The Local Sections included in these districts are as follows:

1. New York.
  2. Boston, Connecticut, Pennsylvania Anthracite, Lehigh Valley, Philadelphia.
  3. Washington, D. C., Southeast, Florida, St. Louis.
  4. Pittsburgh, Cleveland, Ohio Valley, Central Appalachian.
  5. Chicago, Detroit, Upper Peninsula, Minnesota.
  6. Colorado, Wyoming, Utah, Montana, Black Hills.
  7. San Francisco, Nevada, Oregon, Columbia, North Pacific.
  8. Southern California.
  9. Arizona, Southwestern New Mexico, El Paso Metals, Carlsbad Potash, Permian Basin, Southwest Texas, Gulf Coast.
  10. Kansas, Tri-State; Mid-Continent, Oklahoma City, North Texas, East Texas, Delta.
- When a vacancy in the directorate exists from any of these districts, a nomination will be made by a committee consisting of the Chairmen of the Local Sections. It is suggested that they offer a choice of two men, of different Branch interests.

The Committee also suggested that in 1951 the Nominating Committee name not only a President but also a President-elect, and that thereafter only a President-elect be named each year, who would thus have at least one year of service on the Board before becoming President.

The Committee's plan was adopted and the Secretary was directed to draw up the necessary changes in the bylaws to make it effective.

Distribution was authorized to all new Junior Members of the Petroleum Branch of the Robert E. Hardwicke volume on "Antitrust Laws et al vs. Unit Operation of Oil and Gas Pools," through the courtesy of the Seeley W. Mudd Fund. The translation of "Bergwerk- und Probierbuchlein" was added to the list of volumes from which new Junior Members might choose as an added Mudd Fund gift volume. A second edition of "Basic Open Hearth Steelmaking," to be sponsored by the Fund, was also authorized.

Plans for the 1950 Directory of AIME members were discussed. Instead of printing the Directory from the master file cards at Institute headquarters as formerly, postal cards will be sent to all members about March 1, asking them to supply up-to-date information for printing in the Directory. These cards, when returned, will be sent direct to the printer. If cards are not returned, the titles and addresses on record at the Institute will be used instead. Much of the material in the first 115 pages of the 1948 Directory will be omitted, as will the listing of Student Associates.

The Board referred to the Council of Section Delegates the suggestion of the Gillson Committee on Democratization that a grievance committee be appointed to hear all members' complaints, seek out their cause, and do something about their rectification. Subsequently the Council decided that the hearing of grievances was a matter that it might itself assume, and each delegate was asked to be the clearing house for grievances emanating from members in his own local area.

On the day preceding the Board meeting the Council of Section Delegates had an all-day meeting, as a result of which certain matters were brought to the attention of the Board: (1) The Council voted to change its bylaws so that an alternate would have the privilege of voting, provided the Local Section from which he came certified to the Secretary of the Council three weeks prior to the meeting the selection of such alternate. The change was approved. (2) The Council asked for information on the reapportionment of Directors to geographical districts. The Board regarded that this had been answered in the report of the Peirce Committee. (3) A reduction in the initiation fee was recommended in the case of Junior Members applying for change in status to Associate Member or Member. It was suggested that one dollar per year of dues-paying membership be credited, up to a limit of \$10. This was referred to the Admissions and Finance Committees for study and report to the Board. (4) The Council suggested that a committee be formed from the most influential members of the Institute to approach heads of companies from whom advertising in the Institute's journals might be secured. This aroused a great deal of discussion, which brought out the need for additional funds if all of the technical papers of merit presented to the Institute are to be published. A transcript of the discussion will be prepared for the Finance Committee to study.

The Council commended the Institute staff for the excellence of the three new journals, and President Young for his excellent and indefatigable services to the Institute during his term of office.

Following are the new officers of the Council of Section Delegates: Gerald M. Cover, Cleveland Section, chairman; M. I. Signer, Colorado Sec., vice-chairman; T. B. Counselman, N. Y. Sec., secretary; Jack M. Shepherd, East Texas Sec., L. M. Scofield, Minnesota Sec., F. W. Strandberg, Montana Sec., and F. T. Moyer, Washington D. C. Sec., executive committeemen.

Bylaws of the Minerals Beneficiation Division were amended to provide for a symposium committee, and to require securing acceptance of



nominations from the nominees. Bylaws of the new Florida Section were approved, as were changes in the bylaws of several other Sections.

Raul de la Peña informally presented an invitation for the AIME to join with the GSA and the SEG in holding a meeting in Mexico City in 1951, and an invitation was presented by the Boston Section to hold the Annual Meeting of the AIME in Boston in 1953. Action was deferred in each case, the Board expressing its appreciation of the invitations.

A military indoctrination conference will be held in Washington, D. C. in May, and the AIME is invited to send some twenty of its members thereto. Much confidential information as to the military situation, of interest to engineers, will be imparted. Travel expenses to Washington will not be paid. The Secretary's office was asked to receive and forward the names and addresses of all members who would like to participate.

J. Terry Duce and C. Will Wright were approved as AIE representatives on the EJC Committee on International Relations. T. S. Washburn was appointed representative on the Washington Award Commission for a term of two years. Reports on the recent property valuation convention in Lima, Peru, were received by the Board from A. R. Merz and Manuel B. Llosa A., who were AIME delegates.

#### **Officers and Directors Elected at Executive Session**

At the executive session of the Board, held at 5 p.m. Feb. 13, C. V. Millikan was elected vice-

president to succeed D. H. McLaughlin for a one-year term ending in February 1951; Earle E. Schumacher was elected a Director for a one-year term ending in 1951; and Hjalmar W. Johnson was elected a Director for a two-year term ending in February 1952 to succeed William E. Brewster, deceased. Andrew Fletcher was re-elected Treasurer, and Edward H. Robie Secretary, each for a one-year term. The following were named on the Executive Committee: D. H. McLaughlin, chairman; Earle E. Schumacher, vice-chairman; Philip Kraft, E. C. Meagher, and W. M. Peirce. The following comprise the Finance Committee: Erle V. Daveler, chairman; A. B. Kinzel and Gail F. Moulton.

The following dates for meetings of the Board and the Executive and Finance Committees were approved:

March 15, Exec. and Fin. Com., New York

April 19, Board, New York (may be changed to April 21)

May 17, E&F Com., New York

June 21, Board, New York

July 19, E&F Com., New York

August 16, E&F Com., New York

August 29 or 30, Board, Salt Lake City

September 20, E&F Com., New York

October 18, E&F Com., New York

November 15, Board, New York

December 20, E&F Com., New York

January 17, 1951, E&F Com., New York

February 18, Board, St. Louis.



## **SECTION ACTIVITIES**

### **Pittsburgh Section**

The Pittsburgh Section held its Institute of Metals night on Feb. 8 at the Bureau of Mines auditorium. G. J. Donaldson presided; Richard H. Stone was technical chairman. Speaker of the evening was W. M. Ball, Jr., metallurgist and foundry consultant, R. Lavin and Sons, Chicago; his subject, human engineering.

Poor weather held attendance down to 40 but those

who heard Mr. Ball recommend him highly. He presented some interesting views and thoughts on human beings as machines whose behavior must be engineered, developing his point by showing how human behavior is made up of desirable qualities which must be enhanced, and undesirable qualities which must be suppressed in much the same engineering manner that metals are refined and structures or machines are designed and built.

### **FOURTH ANNUAL INSTITUTE OF METALS DIVISION NEW ENGLAND REGIONAL CONFERENCE**

**April 21, 22, 1950, Sheraton-Biltmore Hotel, Providence, R. I.**

Final arrangements are now complete for the New England IMD Regional Conference according to Roger Sutton, general chairman. Plant trips, technical sessions and a dinner are included. Those who wish to take either of the two Friday morning plant trips should make reservations by advising George P. Swift, 53 Galen St., Watertown, Mass., not later than April 14. There is no charge for the plant trip but visitors must register for the Conference in advance by sending the two-dollar (\$2.00) registration fee to Dr.

Swift indicating which trip they wish to take.

The committee would appreciate advance reservations accompanied by checks for both the Conference and the dinner, whether or not you plan to take the trip. The registration fee for the technical session is two dollars (\$2.00), the dinner is four dollars (\$4.00), and a combined price of five dollars (\$5.00) has been established for registration and the dinner. Checks should be made out to George P. Swift. An early response would facilitate plans.

---

**Attention IMD authors—See announcement on p. 622**

---



## PROGRAM

**Friday, April 21, 1950**

**9:00 a.m. Registration, Sheraton-Biltmore Hotel**

**10:00 a.m. Inspection trips:** (Bus transportation available)

Kennecott Wire & Cable Corp., Phillipsdale, R. I.

Washburn Wire Co., Phillipsdale, R. I.

The heating, rolling, cleaning, drawing and melting operations for the manufacture of wire will be observed at these plants.

**2:00 p.m. Technical Session: Formability**

Chairman: K. H. Mairs, Rhode Island State College

"Relation of Mechanical Property Measurements to Formability" by John R. Low, Jr., General Electric Co.

"Composite Metals" by Sidney Siegel, General Plate Division

"Torsion Texture of Copper and Aluminum" by Walter Backofen, Massachusetts Institute of Technology

"Metal Stampings" by Walter Gulliksen, Worcester Pressed Steel Co.

"Production Stamping and Rust Proofing of Gasoline Pump Panels" by Joseph A. Tinsman, Gilbert and Baker Mfg. Co.

**5:30 p.m. Cocktails**

**7:00 p.m. Dinner: Sheraton-Biltmore Hotel**

Toastmaster: H. H. Harris, President, General Alloys Co.

Speaker: Thomas E. Lloyd, Manager of Publications and Editor of Journal of Metals, AIME

"The New England Steel Mill Situation"

**Saturday, April 22, 1950**

**9:30 a.m. Symposium on Fatigue**

Chairman: A. I. Blank, Chase Brass and Copper Co.

"Fatigue and Its Relationship to the Mechanical and Metallurgical Properties of Metals" by George R. Gohn, Bell Telephone Laboratories

"Influence of Surface Processing Upon the Fatigue Strength of Hardened Steel" by Leo Tarasov, Norton Co.

"Fatigue Properties of Chromium Plated Steel" by H. J. Noble, Pratt and Whitney Aircraft Division

"Fatigue in Single Crystals of Iron" by Frank A. McClintock, Massachusetts Institute of Technology

"The Effect of Nitriding on the Fatigue Strength of Stainless Steel" by V. T. Malcolm and Sidney Low, Chapman Valve Mfg. Co.

**12:00 Noon: Luncheon, Hotel Sheraton-Biltmore**



# The Drift of Things . . . . . as followed by *Edward H. Robie*

## A New Manager of Publications

As all employers know, it is one thing to get a good man and quite another to keep him. When Tom Lippert, was engaged by the Institute as Manager of Publications last July under a two-year agreement it was hopefully believed that editorial, advertising, and circulation problems had been transferred to a head capable of solving them. Changes in the editorial content and appearance of the *Journal of Metals* and *Mining Engineering* soon became apparent and by the end of the year advertising was also looking up. In the February issues the first story of Cerro Bolivar—Venezuela's phenomenal new iron mine—made publishing history in the mining field. Then, immediately after the Annual Meeting, Tom was offered a most attractive opportunity in a field completely divorced from publishing—as general manager of the newly organized Titanium Metals Corp. It seemed a chance of a lifetime and the Institute could not well do other than accede to his wish to accept it.

The problem of a successor naturally immediately arose. The possibilities were rapidly surveyed, and the final choice was a man highly recommended by Tom Lippert himself, they having been associated together on the staff of *The Iron Age*. His name is Tom Lloyd, and he brings to the job much of the same experience as his predecessor, though a few years less of it.

Mr. Lloyd began his duties on March 20, as Manager of Publications and Editor of the *Journal of Metals*. For the time being at least his sphere will not include the Dallas office.

A new face also appears in the Petroleum Branch office at Dallas where the vacancy caused by the promotion of Joe Alford to Executive Secretary of the Branch has been filled by Jess Adkins, as editor of the *Journal of Petroleum Technology*.

## Too Much Good Stuff

Every magazine publisher endeavors to strike an appropriate balance between editorial and advertising content. This does not mean that he goes to the extreme of turning down advertising, but rather that as advertising increases he publishes more "reading matter". The problem has been somewhat different with the *Journal of Metals*, which started with only a few pages of advertising and an enormous supply of technical papers, especially in the field of physical metallurgy. For the first year of its publication no brake was exerted on the acceptance of good technical material, but the publishing cost became so large in proportion to the amount received from advertising, subscriptions, and from the dues of members in the Metals Branch that a substantial deficit was incurred. This was especially noticeable because of the new method started a year ago of keeping Branch accounts separate.

For the rest of 1950, a much more rigid editorial selection will be made. Acceptances will be restricted to papers of even higher quality than heretofore. They will be condensed to a greater

degree than in the past, thus making them more readable but without omitting essential information. Reviews of the literature in any paper will be more concise, and information of interest only to neophytes will be eliminated. The same data will not be presented in both graphs and tables. Excessively detailed experimental results will be eliminated, the important points being highlighted. Speculation will not be discouraged, but it must be concise. Only really significant micrographs will be used.

We have outlined some of the factors that authors and editors of technical papers in the *Journal of Metals* have been asked to keep in mind, but the same suggestions apply to all technical papers to be written and published by the Institute. In other words, the meatier they are, and the more readable, the better the chance that they will be published, and the more of such papers that can be published in the course of the year.

## Employment for Engineers

A new book entitled "Employment Outlook for Engineers" has just been issued by the Bureau of Labor Statistics of the U. S. Department of Labor (sold by Superintendent of Documents, Washington 25, D. C., for fifty cents). It should be of much interest to college students, educators, young graduates, and to those who employ engineers—in other words, to a substantial proportion of the AIME membership. It is based on data supplied by some 25,000 engineers who participated in the survey. Engineers Joint Council co-operated.

Principal topics discussed are Employment Trends and Outlook, Earnings, and Occupational Mobility.

In the first sentence of the book we learned something: that engineering is the third largest profession, outranked in size only by teaching and nursing. For men, it is the largest profession. It is also one of the most rapidly growing professions and early in 1948 the number of engineers numbered about 350,000, or about three times the combined membership of the AIME, ASCE, ASME, AIEE, and AICHE. This should encourage the membership committees of our societies. By 1960 the number of engineers is expected to be 450,000. Demand for new engineers for the next few years should be 17,000 to 18,000 a year, but in the decade 1940-50 average graduations in engineering will number 19,000 annually. "The number of persons in training is at record levels and the number of graduates expected from these enrollments should more than supply the demands for engineers in the next several years." Last year 44,000 engineers were graduated, so it is evident why some of them had difficulty in finding jobs. Some 47,000 will graduate this year, so job finding may be even more difficult, but the number may drop to 36,000 next year and to 29,000 in 1952. Even these lower figures seem considerably greater than the demand is likely to be, though it is expected that in five years or so demand and supply will be in



balance. Mention is also made that men with special ability or training in research and design are likely to fare better than the average.

What should the schools do? Raise the standards of entry and use more exacting selection methods. Discourage the marginal student.

As to earnings, they average the highest for those with the greatest amount of formal education. "Masters" do slightly better than "Bachelors" and "Doctors" earn considerably more, but this may be because the better men go after the higher degrees. It seems to make but little difference whether one works for private firms or for the Federal Government, but state and local governments seem to pay less. Top-grade men do exceedingly well in engineering, but an engineering training is no guarantee of success for the mediocre. Unfortunately, 50% of the graduates will always be in the lower half of the class, but raising the level of the entire group will eliminate most of the mediocre.

Engineers do a lot of moving around from one job to another. That is well known as a general principle but here are presented data. AIME headquarters can attest to the fact, for our change-of-address desk processes about half of the membership every year. What is perhaps not so well known is the degree to which engineers change from one kind of engineering to another—such as from mechanical to mining. In 1946 more than 20 percent of all employed engineers were working in a field different from the one in which they were trained. Of mining and metallurgical engineers, 27 percent were working in other fields, the largest shift of any engineering group.

Chemical engineering is growing faster than the others. Civil engineering seems to be showing the slowest growth at present. "Reasonable" growth is expected in mining engineering, including petroleum engineering and metallurgy.

Salaries seemed slightly higher for mining and metallurgical engineers than for any other engineering group, but not greatly so.

### All Except How to Make the Bomb

Latest of the scientific and engineering developments of the century is, of course, that of atomic energy. Automobiles, airplanes, motion pictures, the radio, and lastly television have in turn attracted the interest of many young men, but now those who are anxious to get in on the ground floor of the newest scientific marvel are attracted to the development of atomic energy. Older men, not trained in modern physics, have but slight knowledge of what this is all about. For both young and old who want to do some serious reading on the subject before the weather gets too hot we commend a new book of 228 pages published by the U. S. Atomic Energy Commission entitled "Atomic Energy and the Physical Sciences". (Superintendent of Documents, Washington 25, D. C., 50¢.) It is the best review of developments to date, current work that is going on, where it is being conducted, and who is doing it that we have seen. Fellowships and educational opportunities are included. If you are in the market for an isotope, or want to know how to recognize one when you meet it on the street, this is just what you need.

### April Board Meeting on the 21st

Owing to the annual meeting of the Homestake Mining Co. in San Francisco on April 18, at which President McLaughlin must be present, and the annual meeting of the Canadian Institute of Mining and Metallurgy in Toronto April 17-19, the meeting of the AIME Board of Directors originally scheduled for April 19 has been postponed until the afternoon of Friday, April 21.

### Looking for Suckers

One of our members asks if we can't do something about the flood of brokers' literature emanating from Canada, trying to inveigle people on this side of the line to "invest" in Canadian mining and oil prospects. This has been going on for a long time, so presumably the efforts are not without their reward, to the brokers we mean. Our readers are not the kind that rise to the bait, so there is no use in our warning you, and you, and you. There will always be suckers, and no particular harm would seem to result if some of their money went to Canada, where American dollars are at a premium. A little of it might conceivably actually be used for mining development. The technique of some brokers, however, is to pretend not to ask for money. For instance, here is a letter that starts, "Before I tell you a thing about this amazing proposition, let me say that this letter does not ask you to spend a single penny. Not only that but WE WILL NOT ACCEPT ANY MONEY NOW. All I want from you is an opinion.

"The first thing I am going to ask you is—Would YOU like to see \$5 grow to \$500.00, \$50 grow to \$5,000.00?" (Note the two zeros after the decimal point; it makes the amount look bigger, and perhaps \$5,000.00 might be misinterpreted as \$500,000.) "It has been done, as you will see if you read further in this letter. And there is a mighty good chance right now that it may be done again. Time and again through the 18th and early 19th centuries the world was electrified by the spectacular mineral discoveries in Ontario and Quebec." (We think he means the 19th and 20th centuries, but let that pass, especial in view of the fact that a lot of Americans seem to be a year off in their reckoning on the present half century, which would seem to us to end on Dec. 31, 1950 instead of 1949.) Then the writer goes on to tell something about the property, in this case a lead-silver mine. It is particularly valuable because lead is "the only known practical defense against ATOMIC RADIATION." (Lead-sheathed houses should be particularly necessary when the atomic war breaks, and their construction will require a lot of lead.) We could go on at length, but we feel sure our readers can guess the rest. No money is asked in this particular letter: merely a promise to invest \$50 in 100 shares of stock provided enough additional promises are received to provide "the necessary monies for working capital to commence the mining of its ore reserves", (to say nothing about supplying a bit of capital to the broker). Such letters always mention the name of the engineer who assures a bright future for the property; and you needn't look him up in the directory of his professional society for almost certainly his name won't be there.



---

## Proposed for Membership

---

Total AIME membership on Feb. 28, 1950, was 16,550; in addition 4705 Student Associates were enrolled.

### ADMISSIONS COMMITTEE

E. C. Mcagher, Chairman; Albert J. Phillips, Vice-Chairman; George B. Corless, T. B. Counselman, Ivan A. Given, P. Malozemoff, Richard D. Mollison, and John Sherman.

Institute members are urged to review this list as soon as the issue is received and immediately to wire the Secre-

tary's office, night message collect, if objection is offered to the admission of any applicant. Details of the objection should follow by air mail. The Institute desires to extend its privileges to every person to whom it can be of service but does not desire to admit persons unless they are qualified.

In the following list C/S means change of status; R, reinstatement; M, Member; J, Junior Member; AM, Associate Member; S, Student Associate.

---

### Alabama

Birmingham—Crowder, Robert Scott. (C/S—J-M). Giles, Claud Bryan. (M). Rankin, Frederick Voelker. (M).

### Arizona

Ajo—Cratty, Donald Robert. (C/S—J-M).  
Phoenix—Henderson, Jesse Franklin, Jr. (M).

### California

Fontana—Cherian, Thonipurackal-Varkkey. (J).  
Fullerton—Ferguson, Cedric Keith. (J).  
Laverne—McKee, Verona Isabelle. (M).

Oxnard—Stehle, Carl Vincent. (C/S—J-M).  
Palos Verde Estates—Field, Almeron Johnston. (J).  
Santa Monica—Macleod, Thomas Le Breton, Jr. (J).  
Sonora—Stinson, Walter Allen, Jr. (C/S—J-M).  
Sutter Creek—Buchanan, Edward. (C/S—S-M).

### Colorado

Denver—Conwell, Cleland Neeper. (C/S—J-M). Corbett, Robin James. (J).

### Connecticut

Bridgeport—Flores, Francisco Almeida. (M).



- Florida**  
*Bartow*—Ellis, Edward Jones. (M).
- Georgia**  
*Atlanta*—Nees, John Morgan. (J).
- Illinois**  
*Chicago*—Parker, Howard Alfred. (R, C/S—J-M). Wey-  
 mueller, Carl Robert. (C/S—S-P).  
*East Alton*—Waters, Wade David. (R, C/S—S-M).  
*N. Riverside*—Seery, Frank Edward. (M).  
*River Forest*—Burnham, Charles Thomas. (J).  
*Riverside*—Weir, John Paul. (C/S—S-J).  
*Rosiclare*—Bailie, Harold Emerson. (M).
- Kansas**  
*Garnett*—Austin, Kenneth Oley. (C/S—S-J).  
*Hutchinson*—Blincoe, Edward Irvine. (C/S—S-J).
- Louisiana**  
*Iota*—Risien, Harvey Vineyard. (C/S—S-J).  
*Morgan City*—Wilson, Fred. (J).  
*Shreveport*—Grage, Victor Parker. (C/S—J-M).
- Massachusetts**  
*Brighton*—Collins, William Francis. (M).
- Michigan**  
*Berkley*—Olson, John Henry. (J).  
*Detroit*—Nagler, Charles Arthur. (C/S—AM-M).  
*Willow Run*—Rush, Adron Isaac. (J).
- Minnesota**  
*Duluth*—Spencer, George Herbert, Jr. (M).  
*Mahtomedi*—Collins, Robert C. (J).  
*St. Cloud*—Brammer, Donald James. (C/S—S-J).
- Missouri**  
*Bonne Terre*—Wolf, Edward Hugo. (M).
- Montana**  
*Butte*—Helehan, Thomas Joseph. (AM)  
*Nye*—Toepfer, Peter Herbert. (M).
- New Jersey**  
*Metuchen*—Haskell, Albert Adams, Jr. (R, C/S—S-M).  
*North Plainfield*—Howe, Herbert E. (C/S—J-M).  
*Rahway*—Conard, Clarence Knight. (M).
- New Mexico**  
*Carlsbad*—Hicks, Charles Wilson. (C/S—J-M). Lydick, Robert  
 Lee. (J). Wheeler, Jesse Everett. (M).
- New York**  
*Brooklyn*—Van Hare, George Frederick, Jr. (J).  
*Chappaqua*—Parks, Francis Newton. (C/S—J-M).  
*Crestwood*—Gieseler, Henry Joseph. (J).  
*Douglaston*—Butterworth, Spencer Keeler. (AM).  
*New York*—Cox, Charles Raymond. (M). Fuchs, Warren J.  
 (J). Petrie, Henry Joseph. (C/S—S-J). Wills, Frank. (J).  
*Niagara Falls*—Webb, James Willard. (M).  
*Port Henry*—Sirola, William Matthew. (M).  
*Suffern*—Ettinger, Wolfgang Gerhart. (J).  
*Tahawus*—Wright, Samuel Irwin. (C/S—J-M).
- Ohio**  
*Berea*—Carroll, Walter. (R, C/S—AM-M).  
*Shadyside*—Lang, Emmett Thomas. (J).  
*Shaker Heights*—Kennedy, Ralph Grant, Jr. (M).
- Oklahoma**  
*Enid*—Snyder, Lenard David. (M).  
*Oklahoma City*—Dannenberg, Richard Martin, III. (C/S—  
 S-J).  
*Tulsa*—Barberii, Efrain Emeterio. (R, C/S—S-J). Giles,  
 Frank Stewart. (M).
- Oregon**  
*Portland*—Balusko, Vincent E. (M).
- Pennsylvania**  
*Bala-Cynwyd*—Hanson, Ralph Clark. (M).  
*Bethlehem*—Howells, Edgar Harris. (C/S—J-M).  
*Bradford*—Eckert, Frank Earl, Jr. (C/S—S-J). Kleiner, Abra-  
 ham. (C/S—S-J).  
*Center Valley*—Sawitzke, Edward A. (C/S—J-M).  
*Devon*—Wharton, Heyward Myers. (J).  
*Freeland*—Goepfert, Albert Richard. (C/S—S-J).  
*Harrisburg*—Pennington, Fred Alexander. (M).  
*Montoursville*—Hoyt, Charles Dalburg. (C/S—S-J).  
*Nemacolin*—Fitzwater, Marcellus M., Jr. (C/S—S-J).  
*Norristown*—Giacobbe, John B. (M).  
*Oakmont*—Brennecke, Margaret Ware. (M).  
*Palmerton*—Stuts, George F. A. (M).
- Pittsburgh*—Lamm, Arnold Edward. (M). Little, Samuel  
 Guy. (M). Phillips, William J. (AM). St. Vincent, Frank.  
 (R, C/S—J-M). Snyder, Clifford H. (M). Toering, Albert  
 J. (C/S—J-M).  
*Summit Hill*—Crawford, David Jenkins. (R, C/S—S-M).
- South Dakota**  
*Rapid City*—Carr, Barton Ward. (J).
- Tennessee**  
*Oak Ridge*—Boyle, Edward J. (C/S—J-M).
- Texas**  
*Austin*—Justen, John Joseph. (C/S—S-J). Khayan, Mihran.  
 (C/S—S-J).  
*Beaumont*—Welchon, James Kenneth. (J).  
*Beeville*—Fightmaster, Robert Richard. (M).  
*Corpus Christi*—Edison, James Elmer. (J).  
*Dallas*—Alford, Joe Benge. (AM). Johnson, John Peyton.  
 (M). Monteith, Edward Everett, Jr. (R, C/S—S-J). Will-  
 iams, Nelson Wheat. (C/S—J-M). Wilson, Malcolm Earl,  
 Jr. (C/S—S-J).  
*El Paso*—Ward, Oscar Thurmond. (C/S—S-M).  
*Falfurrias*—Litherland, James George, Jr. (J).  
*Fort Worth*—Walker, James Donald. (M).  
*Gonzales*—Allen, Paul Adolphus, Jr. (J).  
*Hebbronville*—Thomas, Henry Evans. (J).  
*Houston*—Harris, Washington Irving, Jr. (J).  
*Levelland*—Helm, Cyrus Vard. (J). McPhail, Malcolm  
 Charles. (C/S—S-J).  
*Midland*—Cherry, James C. (J). Griffith, Clayton Oliver. (J).  
*Odessa*—Deats, Hollis Wendell. (R, C/S—S-J). Kidd, Thomas  
 Wayland. (J). Showen, Ellsworth Witt. (M).  
*Refugio*—Tonkin, Robert George (C/S—J-M).  
*San Antonio*—Coates, George Hunter. (M).  
*Wink*—Feeser, John Russell. (R, C/S—S-J).
- Utah**  
*Salt Lake City*—Staats, Fred. (M). Walters, George Wilmer.  
 (R, C/S—AM-M).
- Virginia**  
*Big Stone Gap*—Smith, William Clifford. (M).  
*Blacksburg*—Bennett, Allan Vaden. (J).
- Washington**  
*Spokane*—Fergin, George Stanley. (C/S—S-J).
- West Virginia**  
*Beckley*—Gates, Leslie Clifford. (M).  
*Huntington*—Morrow, Oliver Perry. (AM).
- Wisconsin**  
*Wausau*—Smith, Robert Lewis. (C/S—S-J).
- Wyoming**  
*Superior*—Barry, Jack C. (J).
- Alaska**  
*Anchorage*—O'Shea, John Joseph. (M). Saarela, Leo Hjal-  
 mar. (R, C/S—J-M).
- Ontario**  
*Copper Cliff*—Lee, James Edwin. (C/S—J-M).  
*Islington*—Wilson, James T. (AM).  
*Niagara Falls*—Montgomery, Harold Reginald. (C/S—J-M).  
*Ottawa*—Hume, George Sherwood. (M).  
*Toronto*—Bell, Eric Charles. (J).  
*Virginiatown*—Yule, Samuel Cameron. (M).
- Quebec**  
*Montreal*—Dempster, William Eric. (R, C/S—S-J).  
*Mont Joli*—Fairlie, Bertram Murray. (C/S—J-M).
- Mexico**  
*Estacion Moreno, Sonora*—Llamas, Raymondo. (J).
- Chile**  
*Chuquicamata*—Parsons, Edward Wilson. (C/S—AM-M).
- Ecuador**  
*Guayaquil*—Fitzsimmons, Worth Martin. (R, C/S—J-M).
- Venezuela**  
*Caracas*—Gorham, Edwin Lombard, Jr. (M). Moir, James  
 Thomas, Jr. (M).
- England**  
*Birmingham*—Casewell, Edward William George. (C/S—  
 JFA-M).  
*Chester*—Wilson, Robert William. (J).
- Iraq**  
*Basrah*—Buckingham, Clifford Lewis. (C/S—J-M).



# Ductile Titanium

by P. J. Maddex

and

L. W. Eastwood

The authors describe a continuous method of producing titanium, the feasibility of which has been fairly well demonstrated. Magnesium and titanium tetrachloride are used as raw materials for producing titanium or titanium-alloy ingots.

SUFFICIENT information is now available on titanium and titanium-base alloys to indicate that their engineering properties are adequate to insure them a position of importance among construction materials. Some of the properties of titanium and a large number of titanium-base alloys are described in separate papers<sup>1, 2, 3</sup>. However, in addition to having good engineering properties, a structural material must be relatively inexpensive if it is to have widespread use. Although complete data are not yet available on alloys, it is likely that the engineering properties will be favorable for many applications. Consequently, tonnage demand for the alloys will depend largely on production costs. The cost of producing titanium, therefore, becomes of paramount importance.

At the present time, the production of titanium by the reduction of the chloride with magnesium appears to be economically the most feasible. In all probability, this method of producing titanium

will continue to be the most important method. It should be observed at this point that the quality of the commercial titanium sponge produced by the reduction of  $\text{TiCl}_4$  with magnesium, though not so high purity as the product produced by the Van Arkel method, is undoubtedly adequate for most, if not for all, purposes.

The pioneer work on the production of ductile titanium by the reduction of titanium tetrachloride with magnesium has been done by Kroll<sup>4</sup>. Later, this method was further developed by the Bureau of Mines<sup>5</sup>. By their process, the magnesium is melted in a suitable steel reaction chamber after the air has been displaced with an inert atmosphere. The  $\text{TiCl}_4$  is then introduced and the following reaction occurs:



Five to 15 pct of the magnesium, trapped in the sponge, usually remains unreacted. The titanium sponge formed is fairly massive and a considerable quantity of liquid magnesium chloride can be decanted from the reaction product. After cooling, the balance of the reaction product, titanium, magnesium chloride, and unreacted magnesium, is bored from the reaction chamber, ground, and leached in acid until the powdered titanium residue remains. This titanium in powder form contains a large quantity of hydrogen and is most suitable for powder-metallurgy methods of fabrication. Obviously, it is a batch-type process and the product is quite expensive.

In the early work at Battelle, conducted under Air Force Project RAND, a source of titanium was needed to carry out the various aspects of alloy development. Consequently, a small Bu-

---

*The experimental work described in this paper was sponsored by the Air Materiel Command, Wright-Patterson Air Force Base, Dayton, Ohio, under Contract No. W 33-038 ac-21229.*

*P. J. Maddex and L. W. Eastwood are research engineer and supervisor, respectively, Battelle Memorial Institute, Columbus, Ohio.*

<sup>1</sup> C. M. Craighead, O. W. Simmons, and L. W. Eastwood, *Titanium Binary Alloys*, presented to the AIME, February, 1950.

<sup>2</sup> C. M. Craighead, O. W. Simmons, and L. W. Eastwood, *Ternary Alloys of Titanium*, presented to AIME, February, 1950.

<sup>3</sup> C. M. Craighead, O. W. Simmons, and L. W. Eastwood, *Quaternary Alloys of Titanium*, presented to the AIME, February, 1950.

---



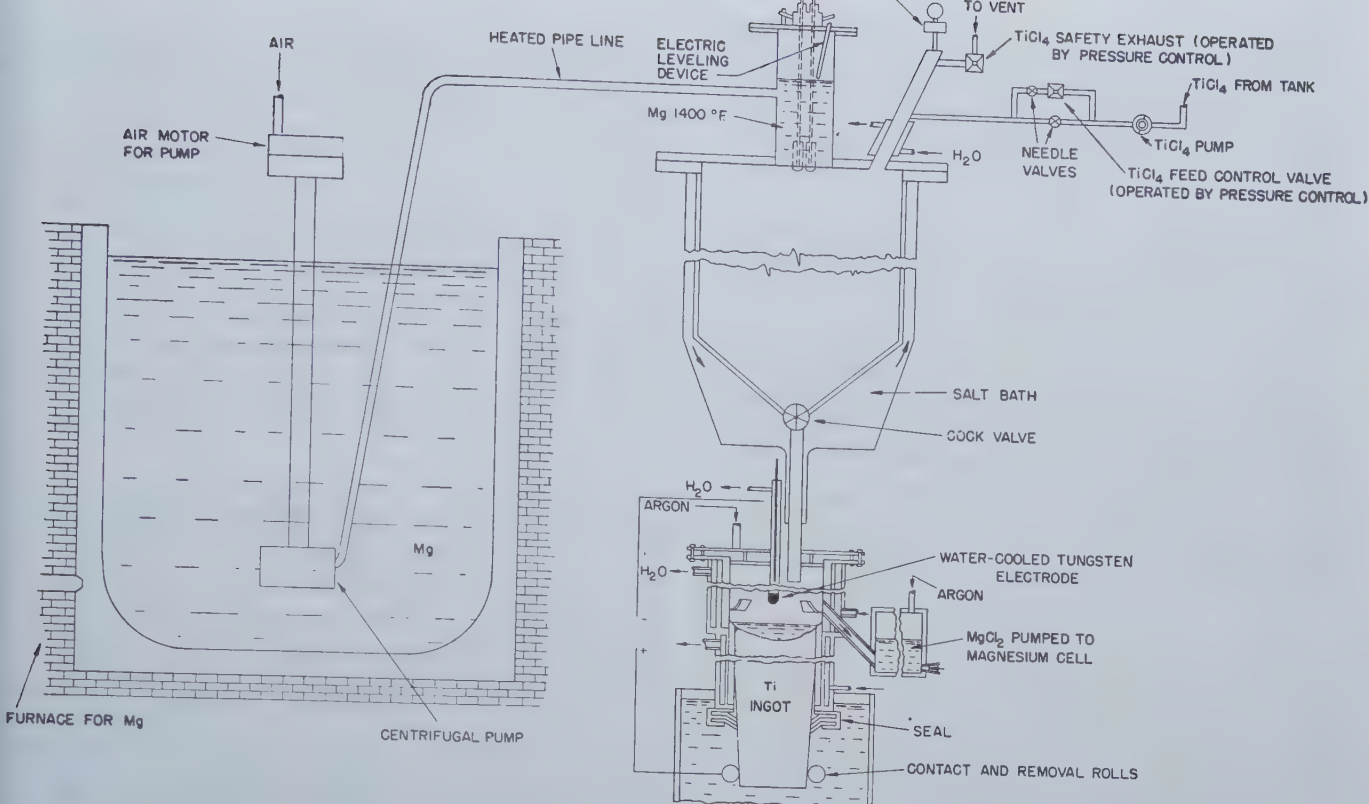


Fig. 1—General arrangement of a proposed pilot plant unit for making continuous titanium ingots from  $\text{TiCl}_4$  and magnesium.

reau of Mines type of unit was developed and operated. It was evident, however, that this process had serious limitations, particularly as a method of producing titanium for the arc-melting process, because of the hydrogen evolution and spattering during melting.

Prior to the time that commercial titanium sponge became available, the Bureau of Mines also investigated the removal of the magnesium chloride and residual magnesium by vacuum distillation. This method of separating the reaction product was also investigated at Battelle.

Briefly, the procedure used at Battelle for the production of titanium and the removal of the magnesium chloride and residual magnesium by vacuum distillation was as follows. The magnesium was placed in a steel bucket which was placed inside of a larger steel reaction chamber, the internal diameter of which was slightly larger than the diameter of the bucket. The air in the reaction chamber was displaced with an inert atmosphere, the magnesium melted, and the  $\text{TiCl}_4$  introduced until the reaction ceased. A large portion of the liquid magnesium chloride was then decanted into a side chamber where it solidified and cooled. The reaction chamber, attached to a suitable vacuum-pump system, was evacuated, and the balance of the magnesium chloride and unreacted magnesium removed by vacuum distillation at a fairly high temperature. After this process was completed, the reaction unit was cooled and opened, the bucket containing the titanium removed, and the process repeated.

While this method for removing magnesium chloride and unreacted magnesium appeared to be somewhat better than acid leaching, the development of this process was not entirely completed because a similar sponge material became commercially available. The immediate need for

another source of titanium no longer existed.

However, experience with the acid-leaching and the vacuum-distillation methods of removing the reaction product clearly indicated that both batch-type methods were expensive and required a very careful technique which had to be duplicated exactly with each repetition of the operation.

**Titanium sponge** produced by the vacuum-distillation process is now available at \$5 per lb. It has been reported<sup>5</sup> that, when the relatively high production of 5,000,000 lb annually is reached, the titanium sponge produced by this process would still sell at \$1.50 per lb. While this represents excellent progress in the production of a relatively difficult metal, the tonnage of the metal used at this price will be rather small. This, in turn, will mean that fabrication processes will be on a relatively small scale and, as a result, it may safely be assumed that the use of titanium will be largely confined to a few military applications where the high cost can be justified. Thus, in 1947, it appeared that a cheaper and more reliable method of producing titanium was very much needed. Two years later, excellent commercial progress has been made but the predicated price, even with high production rates, still indicates the very real need for cheaper methods of producing ductile titanium.

Early in the work on titanium under Air Force Project RAND, a method was conceived by which titanium might be produced continuously, cheaply, and with greater conservation of manpower in time of emergency. Furthermore, it is possible that an even more pure product may be produced than by batch processes. This paper is essentially a progress report on the development of this continuous process, which consists of two parts: (1) the reaction of liquid magnesium,



with  $\text{TiCl}_4$  vapor in a reaction unit to form a reaction product consisting of 11 pct solid titanium and 89 pct liquid magnesium chloride by volume; and (2) the separation of the magnesium chloride and unreacted magnesium by vaporization when the titanium is melted in an arc furnace.

**Features of the process:** Several experimental runs have been made with the reaction chamber. In addition, a considerable amount of work has been done to establish the conditions necessary for the production of titanium ingots in the arc furnace, using a reaction product consisting of magnesium chloride and titanium as the feed material. As a result of this experimental work, briefly described later in this paper, the feasibility of the various aspects of the process has

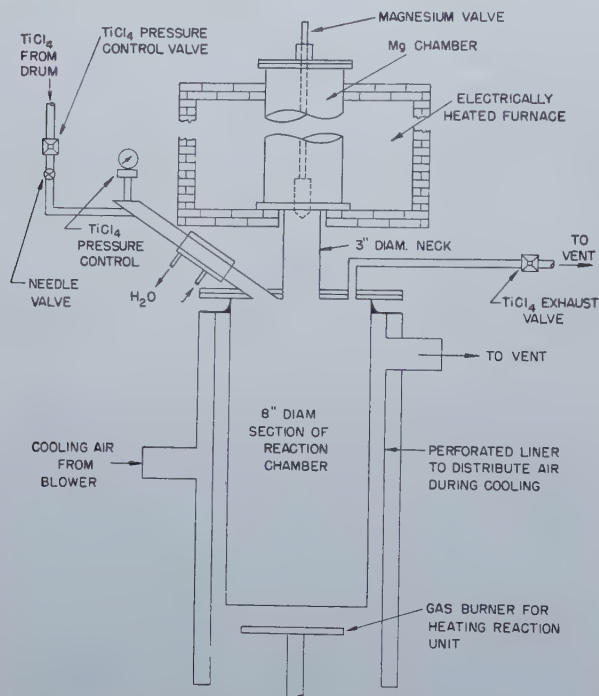


Fig. 2—Detail of valves and orifices.

been fairly well explored. In addition, a fairly clear conception has been reached of the general relationship of the various parts of the apparatus anticipated for pilot-plant operation. This relationship is shown by fig. 1, and the following is a more detailed description of the mechanism of the process.

The reaction unit consists of two essential parts: (1) a means of adding liquid magnesium at a constant rate, (2) a means of adding liquid  $\text{TiCl}_4$  under automatic control to maintain a constant pressure of  $\text{TiCl}_4$  atmosphere. The reaction product, consisting of 89 pct liquid magnesium chloride and 11 pct solid titanium particles by volume, is not accumulated in the reaction unit, but probably will flow continuously through a hole in the bottom of the reaction chamber into the arc-melting furnace below.

The reaction chamber consists of a steel shell into which a stream of magnesium is continuously introduced at a constant rate. Titanium tetrachloride in liquid form is also introduced into the reaction chamber at a rate to maintain a  $\text{TiCl}_4$  atmosphere at a pressure of about 1 to 3

lb above atmospheric pressure. The reaction is exothermic and, therefore, it is unnecessary to heat this reaction chamber after the operation has started. Portions of the reaction chamber adjacent to the reaction zone tend to become hot and those remote from the reaction zone tend to be cold. Unless all portions of the reaction chamber are maintained, preferably, at about  $1350^\circ$  to  $1600^\circ\text{F}$ ., lower chlorides of titanium are formed in the cold zone. As shown in the schematic drawing, fig. 1, it is anticipated that a salt bath circulated around the reaction unit would be the most practical means of maintaining the proper temperature of the reaction-chamber shell. The salt bath would merely be a means of cooling the hot spots and heating the cold regions.

<sup>4</sup> R. S. Dean, J. R. Long, F. S. Wartman, and E. L. Anderson, *Preparation and Properties of Ductile Titanium*, AIME Tech. Pub. 1961, 1946, 13 pp.

<sup>5</sup> *Titanium—Our Next Major Metal*, Product Engineering, November, 1949.

<sup>6</sup> *Air Force Project RAND Report R131, Titanium and Titanium Alloys*, March 15, 1949.

<sup>7</sup> W. J. Kroll, *Production of Ductile Titanium*, Trans. Electrochem. Soc. 1940, 78, 35.

The magnesium is introduced through an orifice  $\frac{3}{64}$  or  $\frac{1}{16}$  in. in diameter, employing a needle valve, the design of which is shown in fig. 2. An argon atmosphere at a pressure of 1 to 2 psi greater than the internal pressure of the reaction chamber is employed over the magnesium melt. By this means, a constant flow of magnesium into the reaction chamber is maintained. In the experimental work, a rate of  $\frac{1}{2}$  to  $1\frac{1}{2}$  pounds per minute, depending on the size of the needle valve, was used.

Under pilot-plant operations, molten magnesium would be pumped to the chamber directly over the reaction unit. Automatic electric leveling devices would be employed to maintain the level of the molten magnesium in the large holding furnace and in the chamber at the top of the reaction unit. Since both the pumping and the leveling devices developed by the Dow Chemical Co. are in commercial operation, this phase of the pilot plant does not require further exploration.

As shown schematically by fig. 1, a large portion of the titanium tetrachloride is introduced through a manually controlled valve into the reaction chamber. Enough titanium tetrachloride is introduced in this manner to supply almost, but not quite, the required amount. The balance of the titanium tetrachloride is shunted through a control valve mechanism which automatically maintains in the reaction chamber a pressure of 2 to 3 lb above atmospheric pressure.

The reaction product probably would not be accumulated in the reaction chamber, but would be permitted to flow through the bottom orifice as rapidly as it forms. This minimizes the possibility of build-up of titanium particles within the reaction chamber and also eliminates the necessity of maintaining a proper bath level. Alternative methods might include techniques for decanting part of the liquid magnesium chloride before it is introduced into the arc furnace.

**The melting unit:** As indicated, the reaction product at a temperature of about  $1600^\circ\text{F}$  flows



continuously into the arc furnace directly below; the titanium is melted, and the magnesium chloride and any unreacted magnesium are volatilized and condensed. This furnace unit is also schematically shown by fig. 1 and consists of three main parts as follows: (1) an arc-melting unit consisting of (a) a water-cooled, bottomless crucible sealed at the top, and (b) an inert water-cooled tungsten or carbon electrode or electrodes to which the negative power terminal is attached; (2) a means for continuously withdrawing the titanium from the bottom of the water-cooled melting chamber; and (3) a means of removing the vapor or condensing the vaporized magnesium chloride and unreacted magnesium and removing the condensate from the melting chamber.

The theoretical quantity of heat to vaporize the magnesium chloride introduced into the arc furnace at 1600°F is less than 1 kw-hr per lb of titanium.

Obviously, the development of a pilot plant for the continuous production of titanium ingots from magnesium and titanium tetrachloride as raw materials is a fairly large undertaking and the job is by no means complete. The aim of the development work to date has been largely to establish the feasibility of various aspects of the pilot plant operation. The present status of these developments is as follows:

The magnesium supply employed to date has been limited to about 25 lb. Consequently, experimental runs have been limited to about 25 min. For pilot plant operation, it would obviously be necessary to have a continuous supply of magnesium. As indicated previously, methods of pumping magnesium and also devices for controlling the level of the magnesium melts are in commercial operation and require no development. This part of the pilot plant, therefore, presents no problem. However, some modification would be required for low rates of magnesium consumption.

The addition of molten magnesium continuously to the reaction chamber in the form of a fine stream presented some difficulties. It is believed, however, that this phase of the problem has been substantially solved. The needle-valve arrangement shown by fig. 2 is made of steel, excepting the probe pin which is made of Nobilium, a cobalt-base alloy. The guide retains the valve rod and probe pin in permanent alignment with the orifice. This is an important detail because it reduces or eliminates the possibility of damaging a rather small-diameter probe pin. Obviously, the resulting valve is an off-on type. The probe pin frees the orifice from any solid material. Magnesium flow is started by merely raising the pin from the orifice.

By the employment of this needle valve, the molten magnesium stream,  $\frac{3}{64}$  or  $\frac{1}{16}$  in. in diam, was added to the reaction chamber without difficulty. It is important that, during the operation of the needle valve, the magnesium is not forced through it at too high a rate. If a high pressure on the magnesium melt is employed, the velocity of the flow of the magnesium is excessive and the time of contact with titanium tetrachloride atmosphere may be too short. It is believed that the method of continuously adding molten magnesium to the reaction chamber has been ade-

quately developed for pilot-plant operation.

Greater capacities than 1 lb of magnesium per minute could be obtained, not by increasing the diameter of the orifice but by making a rectangular slot-type orifice  $\frac{3}{64}$  or  $\frac{1}{16}$  in. wide and the desired length to produce the capacity required. A flat, ribbon-like stream of molten magnesium is better than a round one because of the greater surface exposed to the  $\text{TiCl}_4$  atmosphere. As a matter of fact, the problem of adding molten magnesium to the reaction chamber decreases as the rate of flow increases.

**Control of the pressure of  $\text{TiCl}_4$  atmosphere:** Considerable development work has also been placed on methods of continuously adding liquid titanium tetrachloride to the reaction chamber. In the first place, it is more desirable to add the titanium tetrachloride as a liquid than as a gas. The heat of reaction is about 25 pct lower when a liquid at room temperature instead of a tita-

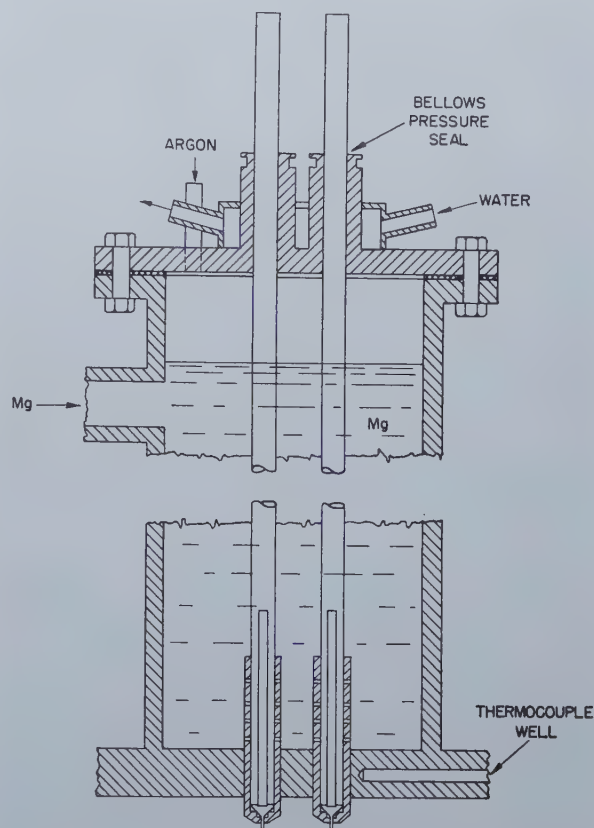


Fig. 3—Reaction unit as originally constructed. All metal parts are steel.

nium tetrachloride vapor is introduced into the reaction chamber. The problem of cooling the reaction chamber is, therefore, lessened by the employment of the liquid.

A successful method of adding the titanium tetrachloride is shown schematically by fig. 1. The titanium tetrachloride is introduced directly from the tanks or drums in which the liquid is shipped. The reaction between the molten magnesium and the titanium tetrachloride atmosphere is not excessively rapid, and the regulation of the pressure of the titanium tetrachloride within the reaction chamber can be maintained at about 2 psi without any perceptible variation. The automatic pressure control equipment



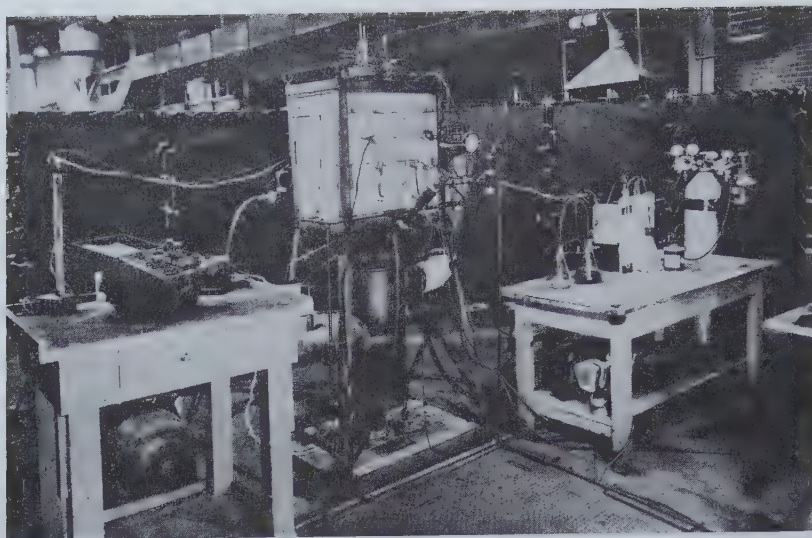


Fig. 4—Continuous reaction unit in operation.

is available commercially. It is believed that this method of adding the liquid titanium tetrachloride is suitable for pilot-plant operation without further development.

**Life of the reaction chamber:** The reaction chamber has been employed for several runs lasting for periods up to 25 min. No difficulties with corrosion or with build-up of titanium on the walls of the reaction chamber have been noted to date. On the other hand, the length of operation has not been sufficient to evaluate these problems fully. It is not anticipated, however, that either of these problems will cause serious difficulty. Since the reaction chamber is essentially a simple steel shell, which could be designed to be easily replaceable, an extremely long useful life is not essential to the success of the operation.

**The furnace unit:** The separation of the residual magnesium and magnesium chloride from the titanium in the arc furnace is an important part of the process. This method obviously has advantages of simplicity and continuity of operation, and provides minimum opportunity for contamination of the titanium. If the upper portion of the refractory-lined arc furnace were operated at 1600°-1800°F, the magnesium and magnesium chloride could be removed as a liquid; at 2600°-2800°F, it could be removed as a vapor.

Ductile titanium ingots have been successfully produced by special melting practices designed to prevent contamination of the metal. One of these methods, developed at Battelle, is electric arc melting in a water-cooled copper crucible. This method has been described in greater detail in another paper<sup>1</sup>. Ingots up to 20 lb have been made and still larger ones could be made by this method. In order to withdraw successfully a continuously made ingot from the melting zone, it is essential that the melt does not wet the material forming the crucible or mold walls. Since titanium melts wet carbon and all other commonly known refractories, they are not suitable for retaining molten titanium for the continuous production of ingots. Fortunately, however, the molten titanium does not wet copper which is kept cool with water. Arc melting in a water-cooled copper vessel, therefore, probably

constitutes the best melting method for the continuous production of ingots. Accordingly, an arc furnace of the type developed on Air Force Project RAND was employed in this experimental work<sup>6</sup>. It was found that titanium in the reaction product, consisting of titanium and magnesium chloride, melted in the arc furnace as satisfactorily as commercial titanium sponge, *provided the magnesium chloride is anhydrous*. For a given set of conditions, the melting rate of titanium in the reaction product was about the same as if titanium alone had been charged to the furnace. If, however, the magnesium chloride was not anhydrous, the arc was unstable and considerable spattering occurred. Obviously, the reaction product made by this process would be completely anhydrous, and, therefore, the feasibility of separating the titanium from the rest of the reaction product was established satisfactorily.

The continuous withdrawal of the titanium ingot from the bottom of the furnace has been accomplished without difficulty. The continuous removal of the magnesium—magnesium chloride vapor or condensate has not been undertaken. Neither has there been any attempt to operate the reaction chamber and melting furnace simultaneously as a unit.

**Experimental work:** The schematic drawing illustrating the relationship of various parts of the first reaction unit is shown by fig. 3. One view of the reaction unit as set up and operated in the laboratory is illustrated by fig. 4. This reaction unit was constructed from steel pipe. The magnesium was melted in a steel chamber placed in an electrically heated furnace. The temperature of the melt was automatically controlled. The valve mechanism, illustrated by fig. 2, was mounted in this melting chamber. The reaction chamber was placed immediately below the needle valve. The melting chamber and the reaction chamber were connected by a short length of 3-in.-diam pipe.

A gas burner was installed at the bottom for heating the reaction chamber, and a light steel manifold was constructed around the reaction chamber to assist in heating it. Air could also be blown through the manifold for cooling if the necessity arose.



All of the titanium tetrachloride was passed through a single orifice and was subject to an automatic off-on control; i.e., when the pressure exceeded about 2 psi, the entire  $\text{TiCl}_4$  flow was cut off automatically; when the  $\text{TiCl}_4$  atmosphere pressure in the reaction chamber dropped to about 1.5 psi, the entire flow was turned on automatically.

This reaction unit operated much more satisfactorily than was anticipated in the beginning. There were only three serious experimental difficulties, which had to be corrected, as follows:

- (1) Because the entire flow of liquid  $\text{TiCl}_4$  was subject to an off-on control, considerable pressure variation occurred in the reaction chamber. Smooth pressure control is necessary to avoid a vacuum if the  $\text{TiCl}_4$  supply is inadequate or excessive pressure if there is too much. A high titanium tetrachloride pressure will blow the vapor into the magnesium orifice, and this may cause clogging from the deposited solid titanium. Consequently, the attainment of smooth pressure control was very essential to the technique of operating the reaction unit.
- (2) The 3-in.-diam neck was too cold since no provision had been made to heat it.
- (3) Portions of the reaction chamber ran too hot, others too cold. Too high a temperature would endanger the equipment, as well as vaporize the magnesium, and might cause a titanium "smoke" to be formed. Too low a temperature caused the formation of lower titanium chlorides.

In order to overcome these difficulties, slight modifications were made in the reaction unit as illustrated by fig. 5. The principal changes which have been incorporated are as follows:

- (1) The main portion of the  $\text{TiCl}_4$  was bypassed through a needle valve manually set to permit enough titanium tetrachloride to almost, but not quite, maintain the pressure in the reaction unit. The balance of the titanium tetrachloride needed was by-passed through an automatic control valve. Approximately 15 pct of the total titanium tetrachloride was passed through this mechanism. Consequently, 85 pct of the titanium tetrachloride flowed continuously, and the off-on control was operated only on 15 pct. As a result, a very uniform pressure of about 2 psi of titanium tetrachloride atmosphere was maintained in the reaction unit. Because the magnesium was introduced roughly at a rate of 1 lb per min, approximately 4 lb of titanium tetrachloride were required per min.
- (2) A burner, not shown by fig. 5, was constructed to heat the neck portion of the reaction chamber. The 3-in.-diam pipe was replaced with a conical-shaped neck, as shown in fig. 5. Provision was also made for air cooling the neck after the reaction had started. The burner at the bottom, for heating the reaction chamber, and the manifold for heating and air cooling, were retained but are not shown in fig. 5.
- (3) Several thermocouples were mounted in the apparatus, as shown by fig. 5, to assist

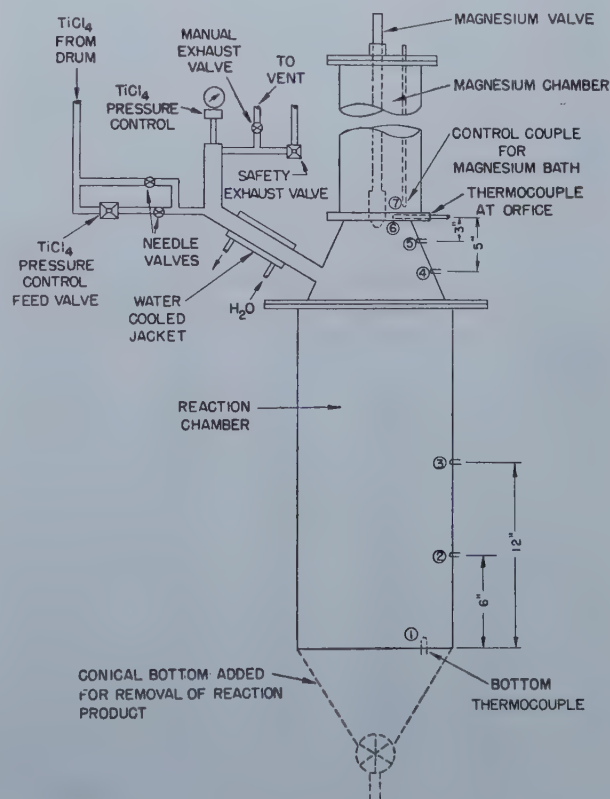
in the control of the temperature of the reaction chamber.

- (4) A conical bottom was attached to the bottom of the reaction chamber to permit the reaction product to flow from the chamber.

A continuous record of the temperatures at the seven different thermocouple locations was maintained, and a typical log is represented by the data in table I. The reaction occurs very smoothly with the generation of some heat. Because most of the magnesium burns in the  $\text{TiCl}_4$  atmosphere near the top of the reaction chamber, some air cooling was necessary in the throat section. As soon as the reaction product reached the bottom of the reaction chamber, continued heating of the bottom became unnecessary. The data shown in table I are for a run made without the conical bottom attached. The reaction product was allowed to collect in the bottom of the chamber. As indicated previously, it is believed that the best way to maintain the uniform temperature of the shell is to place it in a circulated salt bath so that the heat removed from the hot portions of the shell would heat the cold parts.

As indicated in the previous section, the reaction unit is fairly well developed and only requires a continuous magnesium supply for prolonged operation. After this has been done, the problem of corrosion of the reaction chamber or the build-up of titanium can be evaluated. If the reaction product is not accumulated within the reaction chamber, titanium build-up could occur only from the small amount of smoke formed by the reaction of magnesium vapor with the titanium tetrachloride vapor. It might also

Fig. 5—Reaction unit after some modification of the original design shown in Fig. 3. The location of the thermocouples is shown by Nos. 1 to 7.





occur near the discharge if the liquid reaction product contacted the shell of the reaction chamber. Any problem of corrosion or titanium build-up would be a function of the material forming the interior of the reaction unit. This introduces the possibility of using liners to reduce these difficulties should they arise. For example, observation made during the operation of the small Bureau of Mines unit indicated that the reaction product does not adhere to carbon. Other experiments showed that titanium in contact with carbon for 24 hr at 2000°F does not produce any diffusion at the zone of contact.

**Arc stability:** When the first attempts were made to melt the titanium and to remove the magnesium chloride and residual magnesium by arc melting, considerable difficulty was encountered with arc stability. The material employed for the charge was taken from the small Kroll-type reaction unit referred to previously in the paper.

Experimental work showed that a superimposed high-frequency current, higher generator voltages up to 190 v, the employment of argon to sweep the area between the charge and the electrode, and the addition of sodium chloride all improved arc stability. This led to the investigation of a sodium chloride-titanium reaction product instead of a magnesium chloride-titanium reaction product. It was found that the sodium chloride-titanium reaction product melted very readily and smoothly without difficulties or arc instability or spattering. Excepting for the possibility of hydration, the titanium-magnesium chloride reaction product should function as satisfactorily in the arc furnace as sodium chloride and titanium. Consequently, fresh titanium-magnesium chloride reaction product was prepared and a portion of it removed by using a hollow core drill to avoid exposure of the reaction product to the air. When this was done, it was found that the resulting anhydrous product melted very satisfactorily in the arc furnace. For a given set of conditions, the rate of melting titanium in the reaction product appeared to be about the same as when titanium alone was added to the furnace.

The quality of the titanium ingot made by employing the magnesium chloride-titanium reaction product appeared to be about the same as when commercial titanium sponge was employed. Thermodynamic calculations indicated that no appreciable reaction should occur between the titanium and anhydrous magnesium chloride at the temperature of molten titanium. The experimental work confirmed this conclusion.

**Summary:** A continuous method of producing titanium has been proposed and the feasibility of this process has been fairly well demonstrated. The method involves the continuous production of titanium or titanium-alloy ingots, employing magnesium and titanium tetrachloride as the raw materials. Actually, the continuous production of titanium or titanium-alloy ingots involves two steps carried out in a single integrated unit as follows:

- (1) The continuous addition of liquid magnesium to a titanium tetrachloride atmosphere in the reduction chamber where these two reactants unite to form titanium and magnesium chloride at a temperature preferably about 1400° to 1600°F. This reaction product, if there is no residual magnesium, consists of about 89 pct liquid magnesium chloride and 11 pct solid titanium particles by volume. The reaction product probably would not be accumulated in the reaction chamber but would be permitted to run continuously from the bottom directly into an arc furnace where the second step is carried out.
- (2) In the arc furnace, the titanium is melted and the magnesium chloride and unreacted magnesium are volatilized, condensed, and removed from the furnace chamber while a continuously formed titanium ingot is withdrawn from the bottom of the furnace.

**Acknowledgments:** The authors wish to express their grateful appreciation to the Air Materiel Command, Wright-Patterson Air Force Base, who sponsored the work described in this paper and who granted permission for its publication.

TABLE I  
Data For Run No. 6

Time, Min.	Temperature, °F.							TiCl <sub>4</sub> Added, lb Total	Remarks
	Bottom (1)*	6 In. Up (2)	12 In. Up (3)	Lower Throat (4)	Upper Throat (5)	Orifice (6)	Chamber (7)		
3:15 p.m.	1500	1490	1120	1180	1440	1380	1400		
3:25 p.m.	1500	1450	1120	1240	1480	1420	1400		
3:30 p.m.	1420	1440	1120	1260	1500	1420	1400		
1	1800	1360	1120	1260	1660	1380	1400		Run started
2	1900	1320	1140	1260	1840	1400	1400		
3	1850	1400	1160	1300	1880	1420	1400		Heat shut off bottom
4	1820	1420	1200	1320	1900	1440	1400	13	Heat shut off throat
5	1800	1420	1220	1460	1880	1440	1400	18	
6	—	—	—	—	—	—	—		Air cooling turned on throat
7	1860	1500	1540	1660	1740	1440	1400		
8.0	1860	1500	1660	1540	1640	1440	1400		
9.0	1800	1500	1640	1420	1560	1400	1400		
10.0	1720	1520	1600	1400	1500	1420	1400		
11.0	1660	1540	1520	1400	1460	1400	1400	39	
12.0	1580	1560	1460	1340	1520	1400	1400		
13.0	1520	1480	1400	1480	1500	1400	1400		
14.0	1480	1460	1360	1480	1460	1400	1400	49	
15.0	1440	1440	1320	1520	1500	1400	1400	51.5	
16.0	1400	1400	1300	1520	1520	1400	1400		
17.0	1380	1360	1300	1540	1520	1400	1400	55	
18.0	1340	1330	1280	1550	1520	1400	1400	57	
19.0	1320	1320	1300	1540	1500	1400	1400	59	
20.0	1300	1300	1300	1500	1480	1400	1400	61	
21.0	1300	1300	1300	1520	1460	1380	1400	62	Run stopped

\* Numbers represent thermocouple locations shown on fig. 5.



# Use of Adirondack Sinter in Blast Furnaces

by Elmer H. Riddle

JONES & LAUGHLIN Steel Corp's. experience with Adirondack magnetic ores dates back to 1928, when magnetite concentrate from Port Henry, N. Y. was shipped to Aliquippa, Pa. to be used as part of the sintering plant mix. In 1928 through 1930 a high quality sinter was produced from a sinter mix of approximately 60 pct blast-furnace flue dust and 40 pct of this Port Henry magnetite concentrate. Aliquippa No. 5 furnace, for several months during this period, averaged 1200 net tons hot metal daily when using 40 pct of this sinter in the blast-furnace burden. Typical analysis of this sinter was: 60.00 Fe, 10.30 SiO<sub>2</sub>, 1.83 Al<sub>2</sub>O<sub>3</sub>, 1.62 CaO, 0.54 MgO, 0.056 P, 0.35 Mn, 1.7 H<sub>2</sub>O, 0.060 S, and 0.33 C natural.

In 1943-1944 the corporation developed the Adirondack magnetite ore property in the Star Lake, N. Y. vicinity known as the Jones & Laughlin Ore Co. Benson Mines. The mining and beneficiating of this ore and the sintering of the concentrate at the mines are described in the paper "Beneficiation of Adirondack Magnetite," by W. R. Webb, General Sup't., and R. G. Fleck, Plant Sup't., Jones & Laughlin Ore Co., Star Lake, N. Y. That paper appears in the April issue of Mining Engineering Magazine.

The concentrate produced from the Adirondack magnetite ore has a fineness of 100 pct through 20 mesh and 45 pct through 100 mesh. The Adirondack Benson Sinter (hereafter referred to as Benson Sinter) made from this concentrate is shipped by railroad to the corporation's Aliquippa and Pittsburgh blast furnaces. It has a relatively high iron and relatively low silica content compared to the Aliquippa sinter made from hematite classifier and blast-furnace

flue dust. The titania content is 0.95 pct compared to 0.08 pct for the Aliquippa sinter.

It has been evident for a long time that due to the ore supply of the corporation a high percentage of Benson sinter would have to be used in the blast-furnace burden.

In order to determine the effect of a high percentage in the burden on blast-furnace and steel-works operations, a trial run using 50 pct was made on No. 5 furnace from June 1948 through Aug. 15, 1948. The results of this trial run are compared to the results obtained with a normal burden operation designated "reference period."

The trial and the normal burden runs were made in the months which normally have adverse humid weather. No air conditioning of the blast was used. This paper is confined to blast-furnace results. Details of test and comparisons follows:

**Burden factors and iron quality:** All factors affecting iron quality and furnace performance except percentage of Benson sinter were kept as nearly constant as possible in the test period with that of the reference period. This included (1) coke volume per charge and coke quality, (2) percentage of Aliquippa classifier—flue-dust sinter, (3) scrap charged per ton of iron produced, (4) kinds of flux used, (5) high iron temperatures, and (6) iron free of blast-furnace slag.

**Furnace dimensions:** The test furnace, No. 5, has the following dimensions.

Hearth diam .....	26 ft 6 in.
Bosh diam .....	28 ft 9 in.
Stockline diam .....	19 ft 0 in.
Large bell diam .....	14 ft 0 in.
Height .....	90 ft 0 in.
Total volume .....	36,710 cu ft
Working volume (center line tuyeres 6 ft below large bell) ..	31,884 cu ft
Bosh angle .....	84° 54 ft 28 in.
Inwall batter .....	1 5/8 in.
Number of tuyeres .....	18

---

*Elmer H. Riddle is Superintendent Blast Furnace Department, Jones & Laughlin Steel Corp., Aliquippa, Pa.*

*This is a paper to be presented before the Annual Blast Furnace, Coke Oven and Raw Materials Conference, April 10-12, at Cincinnati.*

---



Benson sinter; changes in burden: Except for a nine-day (July 8-16 inc.) all the Benson used was unloaded direct from cars to stockhouse bins. During the nine days it was necessary to use stockpile Benson exclusively because of a temporary shortage of the direct-shipment material. Data for this period were tabulated separately in order that the results could be compared with the direct-shipment period.

**Quality of raw materials:** The chemical analysis of the ores, sinters, fluxes, coke and miscellaneous raw materials are given in table I. This table shows that the Benson sinter was appreciably higher in iron content and lower in silica than the Canby and Newberry ores it replaced in the furnace burden. The alumina content of Benson sinter is higher and its moisture and phosphorus contents lower than the corresponding values for the Lake ores it replaced.

All the coke used was byproduct and was produced at Nos. 3 and 4 coke batteries. Six different grades of coal were used in the coal mix, including from 10 to 14 pct of low-volatile. Several coal changes were made necessary by the availability of the various grades of coal. All the coke is screened at the Byproduct Department; coke over 1½ in. is furnace coke while that between ¾ in. and 1½ in. is termed nut coke and is charged separately into the blast furnace. Throughout the trial period the coke ratio was eleven charges of furnace coke to one charge of nut coke.

As shown by table II, the coke quality did not change greatly throughout the trial. The ash remained nearly constant at about 11.60 pct while the sulphur was about 1.00 pct. The coke strength as indicated by the shatter test also held fairly constant.

The physical nature of the ores and sinters is summarized on table III. The percentage of —20 mesh material in the direct-shipment Benson sinter was under 7 pct whereas the corresponding fines in stockpile Benson was 13 pct. This sinter produced at the Benson Mines is fairly hard-burned and strong, but even with this type of raw material it is evident that the additional handling required to stockpile it resulted in an appreciable increase in the amount of fines. The deleterious effect of this higher percentage of fines in the stockpile Benson will be brought out later in this report.

The relative amounts of the flux materials did not change, consisting of a mixture of 40 pct Blue Stone, 40 pct Millville Dolomite and 20 pct Blair Martinsburg Stone. The dolomite was used to give an MgO content in the slag of about 8 pct.

**Burden and operating data:** Tables IVa, IVb and IVc give the burden sheets for each period. The data given below each burden sheet were calculated from information obtained on the burden sheet. Note that the Fe content in the ore mix is higher and the silica in the ore mix is lower during both high-Benson periods. Notwithstanding this richer mix, during the stockpile Benson period more ore (3294 lb) was needed to produce a ton of iron than during the reference period (3241 lb). This was in part due to the higher flue dust loss during stockpile Benson period, which resulted in a lower iron yield. Another factor was the smaller amount of scrap used during this stockpile Benson period. This

lower yield obtained during the stock pile period indicates the importance of keeping the —20 Mesh material in the charge to a minimum.

The importance of the lower silica value in the direct-shipment Benson period is shown by the lower stone rate per ton of iron produced. This resulted in a lower slag volume, but did not appreciably decrease the coke rate, for reasons to be given later.

Theoretical stock travel rates are given in table V.

Partly as a result of the more open burden, the furnace drove at a faster rate during the direct-shipment Benson period. This was a factor in the higher production figure obtained. Also the equivalent iron in the furnace was greater during this period, mainly accounted for by the higher percentage of Fe in the ore mix.

**Quality of iron produced:** Items dealing with iron quality are shown on table VI. Casting temperatures were considerably higher during the direct-shipment Benson period than during the stockpile period. No temperatures were taken during the reference period but as 40 pct of the silicons were under the specification limit it is probable that they were on the low side.

The higher iron temperatures obtained during the direct-shipment Benson period were associ-

TABLE I  
No. 5 Blast Furnace

Material	Chemical Analysis of Raw Materials (Natural Basis)									
	Nat. Fe Pct	SiO <sub>2</sub> Pct	Al <sub>2</sub> O <sub>3</sub> Pct	CaO Pct	MgO Pct	P Pct	Mn Pct	TiO <sub>2</sub> Pct	H <sub>2</sub> O Pct	
Canby Ore .....	53.88	11.06	0.68	0.37	0.25	0.037	0.24	0.023	7.72	
Newberry Ore ...	51.97	11.55	0.39	0.37	0.25	0.034	0.26	0.030	7.77	
Aliq. Sinter .....	55.67	12.93	1.54	0.38	0.28	0.046	0.28	0.08	6.10	
Benson Sinter ...	64.10	5.38	2.71	0.22	0.24	0.023	0.27	0.95	1.20	
Scrap .....	90.00	.....	.....	.....	.....	0.090	0.50	.....	.....	
Conv. Slag .....	18.33	61.10	1.41	.20	1.00	0.012	7.26	2.22	.....	

Coke Analysis (11/12 Aliq. Byproduct—1/12 Nut)  
(From Ash)

	Fe, Pct	SiO <sub>2</sub> , Pct	Al <sub>2</sub> O <sub>3</sub> , Pct	CaO, Pct	MgO, Pct	P, Pct	TiO <sub>2</sub> , Pct
Coke ...	0.93	5.85	2.66	0.47	0.11	0.014	0.14

Flux Analysis (40 Pct Blue Stone, 40 Pct Dolomite,  
20 Pct Blair Martinsburg)

	SiO <sub>2</sub> , Pct	Al <sub>2</sub> O <sub>3</sub> , Pct	CaO, Pct	MgO, Pct	P, Pct	TiO <sub>2</sub> , Pct
Limestone Mix ...	1.64	0.60	44.07	9.12	0.014	0.021

TABLE II  
No. 5 Blast Furnace  
Coke Quality

	Reference Period	Direct Shipment	Stockpile
Coke ash, pct .....	11.60	11.67	11.54
Coke ash range .....	10.35/13.20	10.70/12.75	10.4/12.4
Coke sulfur, pct ....	1.02	0.98	1.02
Coke sulfur range ...	0.93/1.13	0.88/1.10	0.96/1.09
Shatter test, pct on 2- in. screen .....	—	65.0	66.5
Shatter test range ...	—	59.0/73.2	63.0/69.9

TABLE III  
No. 5 and No. 3 Blast Furnaces  
Representative Screen Analysis of Burden  
Materials (At Stockhouse Bin Chutes)  
Screen Analysis (Cumulative Pct)

Material	No. 5 + 1-in. Tests Mesh		No. 4 Mesh	— No. 20 Mesh	No. 28 Mesh
Benson sinter ....			(+ 0.187 in.)	(– 0.0331 in.)	
Direct shipment..	70	25.72	75.36	6.84	—
Benson sinter ....					
Stockpile .....	9	10.25	53.80	13.00	—
Canby ore					
1948 average ...	3	22.55	51.87	—	31.88
Newberry ore					
6-30-48 .....	1	9.66	43.10	—	37.65
Aliquippa Sinter					
11-8-48 .....	1	14.81	65.91	12.02	—

Benson screen analysis were taken daily from 6-1-48 through 8-25-48.



TABLE IV-A

No. 5 Blast Furnace  
Reference Period, 5/17-5/31 Inclusive  
Burden Sheet, Lb per Ton of Iron Produced

	Burden		Fe		SiO <sub>2</sub>		Al <sub>2</sub> O <sub>3</sub>		CaO		MgO		TiO <sub>2</sub>		Pct	Dry-Fe,
	Pct	Charge	Pct	Charge	Pct	Charge	Pct	Charge	Pct	Charge	Pct	Charge	Pct	Charge	H <sub>2</sub> O	Pct
Canby ....	38	1222	53.88	656	11.06	135.00	0.68	8.30	0.37	4.52	0.25	3.06	0.023	0.24	7.72	58.30
Newberry .	17	546	51.97	284	11.55	62.60	0.39	2.12	0.37	2.02	0.25	1.36	0.030	0.16	7.77	56.20
Benson ....	21	666	64.10	426	5.38	35.80	2.71	18.00	0.22	1.47	0.24	1.60	0.95	6.33	1.20	64.60
Aliq. Sinter	23	758	55.67	422	12.93	97.80	1.54	11.70	0.38	2.88	0.28	2.12	0.08	0.66	6.10	59.40
Conv. Slag.	1	49	18.33	9	61.10	30.00	1.41	0.70	0.20	0.10	1.00	0.49	2.22	1.10		
Total ores.		3241	55.7	1797	11.10	361.20										
Coke .....		1692	0.93	16	5.80	98.00	2.66	45.00	0.47	7.95	0.11	1.86	0.14	2.40		
Stone .....		937	—		1.64	15.40	0.60	5.62	44.07	415.00	9.12	85.40	0.021	0.19		
Scrap .....		234	90.0	211												
Total .....		6104		2024		474.60		91.44		433.94		95.89		11.08		

$$\text{Theoretical iron} = \frac{2024}{0.94} = 2160 \text{ lb}$$

Fe in ore mix = 55.7 pct

SiO<sub>2</sub> in ore mix = 11.10 pct

$$\text{Yield} = \frac{2000}{2160} \times 100 = 92.5 \text{ pct}$$

$$\text{Calculated slag volume} = \frac{433.94 + 95.89}{42.06 + 7.75} \times 0.96 = 1020 \text{ lb per ton of iron.}$$

TABLE IV-B

No. 5 Blast Furnace  
50 Pct Benson Road Sinter, 6/14-7/5+7/17-8/15 Inc.

	Burden		Fe		SiO <sub>2</sub>		Al <sub>2</sub> O <sub>3</sub>		CaO		MgO		TiO <sub>2</sub>		Pct	Dry-Fe,
	Pct	Charge	Pct	Charge	Pct	Charge	Pct	Charge	Pct	Charge	Pct	Charge	Pct	Charge	H <sub>2</sub> O	Pct
Canby ....	24	752	53.88	405	11.06	83.30	0.68	5.10	0.37	2.78	0.25	1.88	0.023	0.15	7.72	58.30
Newberry .	5	165	51.97	86	11.55	19.00	0.39	0.64	0.37	0.61	0.25	0.41	0.030	0.05	7.77	56.20
Benson ....	49	1520	64.10	975	5.38	81.80	2.71	41.20	0.22	3.40	0.24	3.64	0.95	14.40	1.20	64.60
Aliq. sinter	20	616	55.67	342	12.93	79.50	1.54	9.50	0.38	2.34	0.28	1.72	0.08	0.50	6.10	59.40
Conv. slag.	2	74	18.33	14	61.10	45.20	1.41	1.04	0.20	0.15	1.00	0.74	2.22	1.65		
Total ores.		3127	58.2	18	9.90	308.80										
Coke .....		1680	0.93	16	5.89	98.80	2.66	44.60	0.47	7.90	0.11	1.85	0.14	2.35		
Stone .....		850	—		1.64	13.90	0.60	5.10	44.07	376.00	9.12	77.50	0.021	0.17		
Scrap .....		232	90	208	—		—		—		—		—			
Total .....		5889		2046		421.50		107.18		393.18		87.74		19.27		

$$\text{Theoretical Iron} = \frac{2046}{0.94} = 2180 \text{ lb}$$

Fe in ore mix = 58.20 pct

SiO<sub>2</sub> in ore mix = 9.90 pct

$$\text{Yield} = \frac{2000}{2180} \times 100 = 92 \text{ pct}$$

$$\text{Calculated slag volume} = \frac{393.18 + 87.74}{40.22 + 8.30} \times 0.96 = 950 \text{ lb per ton of iron.}$$

TABLE IV-C

No. 5 Blast Furnace  
50 Pct Benson Stockpile Sinter, 7/8-7/16 Inclusive  
Burden Sheet, Lb per Ton of Iron Produced

	Burden		Fe		SiO <sub>2</sub>		Al <sub>2</sub> O <sub>3</sub>		CaO		MgO		TiO <sub>2</sub>		Pct	Dry-Fe,
	Pct	Charge	Pct	Charge	Pct	Charge	Pct	Charge	Pct	Charge	Pct	Charge	Pct	Charge	H <sub>2</sub> O	Pct
Canby ....	24	780	53.88	420	11.06	86.50	0.68	5.30	0.37	2.88	0.25	1.95	0.023	0.16	7.72	58.30
Newberry .	6	180	51.97	94	11.55	20.80	0.39	0.70	0.37	0.67	0.25	0.45	0.030	0.05	7.77	56.20
Benson ....	49	1610	64.10	1040	5.38	86.80	2.71	43.50	0.22	3.54	0.24	3.86	0.95	15.30	1.20	64.60
Aliq. sinter	19	640	55.67	356	12.93	82.60	1.54	9.88	0.38	2.44	0.28	1.80	0.08	0.51	6.10	59.40
Conv. slag.	2	84	18.33	15	61.10	51.00	1.41	1.18	0.20	0.17	1.00	0.84	2.22	1.86		
Total ores.		3294	58.70	1925	9.95	327.70										
Coke .....		1804	0.93	17	5.77	104.00	2.66	48.00	0.47	8.50	0.11	1.98	0.14	2.52		
Stone .....		962	—		1.64	15.80	0.60	5.76	44.07	424.00	9.12	87.80	0.021	0.19		
Scrap .....		225	90	202	—		—		—		—		—			
Total .....		6285		2144		447.50		114.32		442.20		98.68		20.59		

$$\text{Theoretical Iron} = \frac{2144}{0.94} = 2280 \text{ lb}$$

Fe in ore mix = 58.70 pct

SiO<sub>2</sub> in ore mix = 9.95 pct

$$\text{Yield} = \frac{2600}{2280} \times 100 = 88 \text{ pct}$$

$$\text{Calculated slag volume} = \frac{442.20 + 98.68}{40.00 + 8.45} \times 0.96 = 1070 \text{ lb per ton of iron.}$$

TABLE V

No. 5 Blast Furnace  
Theoretical Stock Travel Rates

Per Ton of Iron Ores and Sinters	Lb	Density Cu Ft	Normal Lb	Burden Cu Ft	50 Pct Road Lb	Sinter Cu Ft	Stockpile Lb	Sinter Cu Ft
Canby .....		150	1222	8.13	752	5.02	780	5.20
Newberry .....		140	546	3.90	165	1.18	180	1.28
Benson sinter .....		122	666	5.48	1520	12.50	1610	13.20
Aliq. sinter .....		120	758	6.32	616	5.14	640	5.32
Total ores and sinter .....			3192	23.83	3053	23.84	3210	25.00
Stone .....		100	937	9.37	850	8.50	962	9.62
Coke .....		31	1692	54.60	1680	54.30	1804	58.20
Scrap .....		100	234	2.34	232	2.32	225	2.25
Gross total .....			6055	90.14	5815	88.96	6201	95.07
Less flue dust (Est.) .....		110	186	1.69	213	1.93	347	3.15
Net Total .....			5869	88.45	5602	87.03	5854	91.92

Iron prod., tons per day.....	1215	1260	1137
Cu ft material processed per day...	107000	110000	104000
Working vol, cu ft.....	31882	31882	31882
No. of fillings per day.....	3.36	3.45	3.25
Theoretical travel time, hr.....	7.13	6.96	7.40
Working height, ft.....	66	66	66
Theor. travel rate, ft per hr.....	9.25	9.50	8.92
Equivalent iron in fee. tons.....	360	366	350



**TABLE VI**  
**No. 5 Blast Furnace**  
**Blast Furnace Iron Quality**

	Reference Period	Direct Shipment	Stockpile Benson
Casting temperature, °F (avg.)	—	2710	2688
Casting temperature, pct below 2700°F	—	28.0	61.4
<b>Iron Analysis</b>			
(A) Silicon, pct under 1.20	40.0	24.2	11.1
Silicon, pct between 1.20/1.50	45.0	57.9	77.8
Silicon, pct over 1.50	15.0	17.9	11.1
(B) Sulfur, pct under 0.030	25.0	52.9	40.0
Sulfur, pct under 0.040	60.0	86.0	60.0
(C) Manganese, pct under 0.40	10.0	30.0	0
Manganese, between 0.40/0.50	80.0	62.0	98.0
Manganese, over 0.50	10.0	8.0	2.0
(D) Si/Mn ratio, under 3.0	70.0	36.9	44.4
Si/Mn ratio between 3.0/3.5	16.0	34.7	48.9
Si/Mn ratio, over 3.5	14.0	28.4	6.7
(E) Phos., daily average	0.071	0.068	0.070
Phos., range	0.068/0.075	0.058/0.092	0.065/0.071
(F) Titanium, daily average	—	0.21	0.21
Titanium, range	—	0.15/0.30	0.17/0.27
<b>Slag Analysis, Pct</b>			
SiO <sub>2</sub>	38.00	36.83	37.24
SiO <sub>2</sub> range	36.30/41.40	35.28/38.86	36.28/38.20
Al <sub>2</sub> O <sub>3</sub>	11.44	12.94	12.64
Al <sub>2</sub> O <sub>3</sub> range	10.60/12.10	12.03/14.20	12.22/13.22
CaO	42.06	40.22	40.00
CaO range	38.10/44.10	37.70/42.50	39.00/41.00
MgO	7.75	8.30	8.45
MgO range	6.92/8.50	7.30/9.58	7.43/9.58
S	1.30	1.38	1.40
S range	1.10/1.46	1.26/1.56	1.30/1.48
MnO	—	0.30	0.33
MnO range	—	0.15/0.41	0.26/0.44
TiO <sub>2</sub>	—	1.02	1.04
TiO <sub>2</sub> range	—	0.83/1.17	0.90/1.10
FeO	—	0.26	0.26
FeO range	—	0.17/0.40	0.18/0.33
Base/Acid Ratio ( $\frac{\text{CaO}+\text{MgO}}{\text{SiO}_2}$ )	1.31	1.32	1.30
Base/Acid Ratio range	1.11/1.41	1.20/1.44	1.24/1.36

**TABLE VII**  
**No. 5 Blast Furnace**  
**Blast Furnace Performance and Practice**

	Reference Period Reg. Burden	50 Pct Road Benson	Diff. Am't.	Stockpile 50 Pct Benson	Diff. Am't.
<b>PRODUCTION:</b>					
Iron, tons per day	1215	1260	+45	1137	—78
Scrap, tons per day	141	146	—	128	—
Adjusted iron, tons per day (—¼ scrap)	1109	1151	+42	1041	—68
D.C. flue dust, lb per ton iron	167	134	—33	175	+8
Total flue dust, calculated (2/3 of iron loss)	213	240	—	373	—
Iron yield, pct	92.5	92	—	88.0	—
Calculated slag, lb per ton iron	1020	950	—70	1070	+50
<b>CONSUMPTION:</b>					
Coke, lb per ton iron	1692	1680	—12	1804	+112
Adjusted coke, lb per ton	—	—	—	—	—
Iron	1860	1860	0	2000	+140
Flux, lb per ton iron	937	850	—87	962	+25
Adjusted flux, lb per ton iron	1055	956	—99	1070	+15
<b>PRACTICE:</b>					
Air blown, cu ft per min	75967	78800	+2833	78178	+2211
Blast temperature, °F	1004	1067	—	1024	—
Blast temperature, range	1000/1120	1000/1220	—	900/1100	—
Moisture in blast, grains per cu ft	4.32	5.89	—	6.32	—
No. charges per day	199	210	+11	204	+5
Burden ratio, avg	1.98	1.92	—0.06	1.88	—0.10
Wind cuts, No. per day	0.93	0.47	—	0.67	—
Wind cuts, min per day	312	124	—	161	—
Furnace slip, No. per day	2.2	1.8	—	2.0	—

ated with the highest percentage of O.K. sulphurs, 86 pct of the casts being under 0.040 pct sulphur. The average slag basicity, as measured by the base/acid ratio, was nearly constant for all three periods. Its average value (1.31) indicates a slag slightly on the lean side.

During the direct-shipment Benson period over 63 pct of the casts were over 3.0 silicon/manganese ratio. It was necessary to maintain this high ratio, together with high iron temperatures and

clean iron, in order to avoid sloppy blows at the converting mill, especially when blowing this higher titanium iron.

Titanium values for both high-Benson periods was 0.21 pct in the iron and a little over 1.00 pct titania in the slag. Theoretically, variations in the titanium should parallel the variations in the silicon content of the iron. That is, given a constant weight of titania in the charge, high titanium values in the iron are favored by high



hearth temperatures and a lean slag. As was expected, good correlation was found between iron titanium, iron silicon and iron temperature, as shown by the following table.

Titanium Range,	Average Ti, Pct	Average Si, Pct	Average Temperature, °F
0.15/0.19	0.176	1.21	2690
0.20/0.22	0.212	1.34	2710
0.23/0.25	0.238	1.44	2728
0.26 plus	0.272	1.58	2733

The percentage of reduction of titania averaged about 36 pct during the high-Benson periods. This high-titanium iron appeared to be more fluid than iron of the same chemical analysis (except for titanium) and physical temperature produced when the furnace was on a normal burden. Therefore it appears that iron fluidity is increased by an increase in its titanium content, at least in the 0.16 to 0.28 pct titanium range.

**Blast furnace performance and practices:** Iron production is proportional to the product of the Burden Ratio x pct Fe in the ores x No. of charges per day. Therefore iron production should be increased by an increase in the richness of the ore mix and with an increase in the driving rate of the furnace as measured by the number of charges per day. It should also increase when the silica content in the ore mix decreases, as a decrease in slag-forming material should enable the furnace to carry a heavier burden (higher burden ratio) due to the decreased slag volume. Also the coke rate would be expected to vary directly with the slag volume and inversely with iron production.

As shown by table VII, during the direct-shipment Benson period there was a 42 tons per day (3.8 pct) increase in the adjusted (iron production adjusted to a zero scrap in the charge basis) iron production over the reference period. The two factors favoring this increase in production were:

- (1) A richer burden, as indicated by the 2.5 pct increase in the Fe content of the ore mix, as the Benson sinter was richer in Fe than the ores it replaced.
- (2) An increase in the driving rate of the furnace as indicated by the 5.5 pct increase in the number of charges per day. This increased rate of stock travel is also shown by table V, for the theoretical travel rate during the direct-shipment Benson period was 9.50 ft per hr as compared to 9.25 ft per hr during the reference period. This faster driving was partly due to the more open burden obtained when using the higher percentage of sinter. This permitted an increase in the wind blown by 2833 cfm, at the same time lowering flue-dust loss and obtaining more smooth furnace operation, as shown by the reduction in wind cuts and furnace slips.

A factor acting to limit the amount of increased iron production was the 0.06 point drop in the burden ratio. This lowering burden ratio

during the direct-shipment Benson period occurred in spite of the 70 lb per ton decrease in the slag volume. This partly explained by the lower hearth temperature during the reference period and partly by the probable decrease in furnace thermal efficiency during the higher sinter period. Another angle on the same problem is the constant adjusted coke rate during both the reference and the direct-shipment Benson periods, although the slag volume was considerably lower during the latter period. As no top-gas analyses were obtained, there is no direct evidence that the percentage of CO<sub>2</sub> in these gases showed a decrease during the higher sinter periods, thus there is no proof that this Benson sinter is harder to reduce than the ores it replaced in the furnace burden.

The 68-ton-per-day drop in adjusted iron production and 4.5 pct decrease in yield which occurred during the stockpile period is an example of the unfavorable results to be expected when there is an increase in the amount of fines in the sinter. The percentage of —20 Mesh material in the stockpile Benson was 13 pct, about twice the percentage in the direct-shipment Benson. This shows the importance of careful handling of the sinter and of keeping the number of handlings to a minimum.

**Conclusions:** This experiment indicated iron quality can be satisfactorily maintained when operating with 50 pct Benson sinter and 20 pct Aliquippa sinter in the burden. Iron chemistry (except for titanium) and physical temperatures can be as readily controlled.

In respect to blast furnace performance, the higher sinter burden gives increased iron production, less flue-dust loss and a lower slag volume. The increased production is caused by its higher Fe content and by the more open burden it creates. This more open burden enables more wind to be blown without increasing the flue-dust loss. This statement only holds for Benson sinter of good physical condition, for if the —20 Mesh materials exceeds 8 to 10 pct this favorable condition is eliminated.

More investigational work is needed to show why there was not a marked decrease in the coke rate when the furnace was on the higher sinter burden, as would be expected from the lower slag volume. Whether the thermal efficiency of the furnace was decreased due to an increased amount of direct reduction when on the higher sinter burden can only be accurately determined by continuous sampling of the top gas to obtain its CO and CO<sub>2</sub> content.

This test and subsequent operating investigations indicate that a high blast temperature can be used when operating on a high percentage of Benson sinter in the burden. There are also many indications that a firm and strong porous sinter with its minimum number of fines is preferable to a soft porous sinter containing many fines, even with the sacrifice of reductibility. This seems particularly true at Aliquippa where high volumes of wind are blown. It is not necessary to point out the severe penalty encountered if any physical or chemical change is made in the sinter or any other material charged in such large amounts.



# Blowing Out

## Carbon-Hearth Furnaces

by George D. Sells

ON Feb. 10, 1947, the No. 1 blast furnace of the Pittsburgh Steel Co. was blown in. The 19-ft diam furnace had a new carbon-block hearth and a complete carbon tap hole.

The hearth walls were built of two layers of carbon blocks. The inside ring was constructed of 15-in.-thick blocks, and the outer ring of 10 $\frac{1}{4}$ -in. blocks with a carbonaceous cement bond between the two layers. The outer carbon hearth was further surrounded by a carbonaceous cement and a water-cooled jacket composed of 23 cooling staves. The hearth was 8 ft, 9 in. high, extending to the center line of the cinder notch.

The bottom included inside the hearth jacket was divided into three layers. The bottom layer was composed of 20-in. clay blocks and had a 24-

ft, 3-in. diam. On this base, two layers of 22 $\frac{1}{2}$ -in.-thick carbon block were placed, producing a total carbon thickness of 45 in. Carbonaceous cement was used between the blocks in each layer, but none was used between the layers.

*George D. Sells is Superintendent, Blast Furnace, Pittsburgh Steel Co., Monessen, Pa.*

*This is a paper to be presented before the Annual Blast Furnace, Coke Oven and Raw Materials Conference, Cincinnati, April 10-12.*

On April 16, 1949, preparations were made to blow out the No. 1 furnace due to business conditions. The furnace had operated continuously with only minor delays for approximately two

### BLOW OUT SCHEDULE FOR No. 1 BLAST FURNACE — APRIL 17TH, 1949

Actual Time After Starting of Nut Coke in Furnace		Wind Schedule		Heat, Deg. F	Stock Gas		Water Sprays in Furnace		Cast Analysis	Slag Analysis	Remarks
Hr.	Time	Rev.	C.F.		Line, Ft.	Press., In.	On 1-2	Off 1-2			
0	12:10 a.m.	65	48,100	250	8	6	0	2			Started nut coke here
1	1:10 a.m.	65	48,100	230	8	6	0	2			
2	2:10 a.m.	65	48,100	210	8	6	0	2			
3	3:10 a.m.	57	42,180	370	8	7	0	2			
4	4:10 a.m.	57	42,180	360	8	7	1	1			
5	5:10 a.m.	57	42,180	330	8	8	1	1	Si. 1.52 S. 0.040 Mn 1.84	SiO <sub>2</sub> 35.82 Al <sub>2</sub> O <sub>3</sub> 13.66 S 1.60	Tonnage 105.50 Nut coke all in
6	6:10 a.m.	57	42,180	300	8	10	1	1			Regular charges in Bells empty
7	7:10 a.m.	52	38,480	200	12	6	2	0			
8	8:10 a.m.	52	38,480	150	23	5	2	0			
9	9:10 a.m.	47	34,780	400	27	8	2	0			
9	9:20 a.m.	47	34,780	460	34	10	2	0			
9	9:35 a.m.	47	34,780	380	36	9	2	0			
9	9:45 a.m.	47	34,780	350	36	9	2	0			
10	10:10 a.m.	47	34,780	350	37	8	2	0			Not moving Not moving Not moving Checked furn- ace down
10	10:30 a.m.	47	34,780	340	37	8	2	0			
10	10:45 a.m.	47	34,780	320	37	9	2	0			
10	10:55 a.m.	47	34,780	320	39	9	2	0			
11	11:10 a.m.	47	34,780	350	40	10	2	0			
11	11:35 a.m.	47	34,780	350	42	11	0	2	Si. 1.60 S 0.025 Mn 1.92	SiO <sub>2</sub> 32.20 Al <sub>2</sub> O <sub>3</sub> 12.11 S 2.04	Tonnage 67.90 Furnace down. Wind off.
12	12:10 p.m.	Blow engines stopped				49	0	0			One-half turn on valve
13	1:30 p.m.	Bells opened						0			
14	2:30 p.m.	Sprays turned on top						2			



years and two months during which time it produced 508,543 net tons of iron.

On April 16, 1949, at 12:10 a.m., all scrap turnings, openhearth cinder and 600 lb of flux were taken off. Regular filling was continued until 12:10 a. m. on April 17, at which time charging of 400,000 lb of nut coke was started. This coke was followed by four regular charges in which the regular coke was replaced with nut coke. This stock was well quenched with water. The top heat was further regulated by means of two 40-lb-pressure water sprays placed in the top of the furnace. After this material was charged, the bells were left empty and closed. The wind at this time was 42,180 cf and the stock line at 12 ft, 6 in.

One hour before the last cast, at which time the furnace was blown down to 37 ft, 9 in., with a wind of 34,780 cf, the goggle valve to the boiler house was closed and the steam was put in all gas lines, dust catchers and gas washers. The bleeders were opened sufficiently to keep the pressure normal. Immediately before casting, the wind was 34,780 cf and the stock line was 42 ft, 11 in.

After the last cast, at which time the stock line was 49 ft, the blow pipes were dropped, the tuyeres were plugged, and both bells were opened. Two hours later, the water was turned one-half turn on top spray. One-half turn was made every 30 min until the valve was completely open. The

sprays were left fully open for 8 hr at which time water began to show at the tuyeres and the cinder notch. The water sprays were then turned off. The No. 2 and No. 11 coolers were removed, chutes were placed and washing began. Washing continued for 8 hr in which time all of the nut coke and stock were removed. Six hopper cars of hot nut coke and stock had been removed.

The furnace stack was then cleaned and the material below the tuyere line was removed. The furnace was finally cleaned to a depth of 18 in. below the tap hole and an examination of the hearth was made.

The tap hole was in good condition and appeared to be approximately the same as when it was originally put in. Forty-two inches of carbon could still be seen in the tap hole.

No salamander was encountered two feet below the center line of the iron hole which was as far as the hearth was dug out.

The hearth walls were uniformly covered by a hard formation approximately 7 in. thick. This formation was not removed except for a small section over the iron hole. This exposed area showed a few spots of disintegrated carbon.

A horizontal crack was found at the junction of the carbon hearth and the ceramics in the hearth. This would indicate that a carbon hearth should extend higher than the center line of the cinder notch.

---

by H. P. Saxer

---

NO. 1 blast furnace was blown in on Feb. 5, 1948 following a complete relining that included the installation of a carbon hearth of rather conventional design. The carbon-hearth blocks consisted of two rows of carbon each 22½ in. thick or a total of 45 in. of carbon in the hearth and two courses of vertical carbon blocks each 12 in. thick with a 2-in. space between the blocks filled with a rammed carbon cement. The vertical blocks were left 4 in. from the hearth-cooling segments, this space being filled with a tamped mixture of tarred coke breeze. One exception was the use of small 4x2x9-in. carbon brick in place of the tarred coke breeze around the tapping hole section which was of all-carbon construction.

Due to decreased iron requirements and a desire to investigate the condition of the carbon hearth, it was decided to blow out this furnace. This was done on June 15, 1949—the furnace having produced a total of 379,273 net tons of high-manganese basic iron.

---

*H. P. Saxer is Sup't. of Blast Furnaces, Jones & Laughlin Steel Corp., Pittsburgh. This is a paper to be presented before the Annual Blast Furnace, Coke Oven and Raw Materials Conference, Cincinnati, April 10-12.*

---

The blow-out procedure followed was the same as that used in blowing out ceramic-lined hearths with the exception that care was taken to avoid the use of excessive amounts of water.

Prior to starting the actual blowing-out opera-

tion, a 40-skip coke blank was charged. Upon completion of the coke blank, which coincided with the completion of a cast, the wind was taken off the furnace while four 1½-in. water spray pipes were inserted in test rod holes and a hand test rod installed in place of the automatic gage rod. The large bell was closed and sealed with 20,000 lb of ore, the small bell kept open.

Upon resumption of operations, water was added through the spray pipes and wind reductions made to control top temperatures under 600°F. During the entire blowing-out period the furnace was isolated by means of a goggle valve, the gas escaping out of the furnace bleeders. Top temperatures were taken from thermocouple installed in one of the furnace uptakes.

The furnace was blown down about 50 ft below the closed large bell or to a point about 10 ft above the mantle when the last cast was made, at which time the use of water was discontinued. Following the last cast and after conventional procedures, the blow pipes were dropped and the tuyeres plugged lightly with fire clay. The furnace was allowed to sit for about 4 hr after which gradual increasing amounts of water was added through the spray pipes. As soon as water appeared at the tuyeres, the use of water was stopped. After an interval of about 2 hr water again was added until it was in evidence at the tuyeres. This procedure was followed for about 12 hr, there being no evidence of water at the tapping hole at any time. Following this procedure the furnace was raked out to tuyere level and shoveled out from tuyere level to the top of the hearth.



# European Research in Physical Metallurgy

by B. D. Cullity

**I**N a postwar Europe beset with major economic and political difficulties, scientific research is showing exceptional vitality. Research in all the pure sciences, and in physical metallurgy, is slowly regaining its prewar stature both in quantity and quality and, in a shortage-ridden economy, there appears to be no lack of the one basic requirement for scientific progress: ideas.

The present article is an attempt to make a rapid survey of the present situation. It is based on notes made by the writer during two years residence in Europe, 1948 and 1949, the former year being spent in metallurgical research at the Ecole des Mines in Paris and the latter in scientific liaison for the London branch of the Office of Naval Research, U. S. Navy. Geographically, this survey is limited to Western Europe and

physicists as well as metallurgists are active. Interest in the former subject is reflected in numerous studies of creep and in theoretical work on dislocation theory.

The University of Birmingham has the largest Department of Metallurgy among British universities. Headed by D. Hanson, this department is divided along two main lines, industrial metallurgy and fundamental physical metallurgy. Research in the latter field is directed by G. V. Raynor and is concentrated mainly on the theory of alloy structures and the mechanism of deformation. The work on alloy structures aims at the prediction of specific features of an unknown equilibrium and at a more fundamental understanding of the peculiarities of known phase diagrams. The experimental work involves the determination of phase equilibria to check theoretical predictions and the measurement of quantities such as vapor pressure and electrode potential for use in thermodynamic calculations. Ternary and quaternary alloys are now being investigated with special attention to the role played by transition elements. In the field of plastic deformation, considerable work is being done on dislocation theory and its experimental verification and on x-ray diffraction studies of cold worked metal.

Investigations of interest to metallurgists are being carried out in several departments of Cambridge University:

(1) In the *Department of Metallurgy*, directed by G. W. Austin, work is in progress on a large variety of problems. Internal oxidation is being studied, with particular attention to the hard-

---

*B. D. Cullity is Assistant Professor of Metallurgy, University of Notre Dame, Notre Dame, Ind.*

*All statements made in this article are the responsibility of the writer and not of the Office of Naval Research.*

---

only basic, rather than applied, research in physical metallurgy is considered. It is thus confined to the work of university laboratories, research institutes and the like.

**England:** There is a greater concentration of scientific activity in England than in any other country of Europe. In physical metallurgy, the mechanism of plastic deformation and the electronic theory of alloys are popular fields of investigation, in which a considerable number of



ening effects produced; determinations of stress-strain diagrams of internally oxidized specimens will be made shortly. A spherical-mirror reflecting microscope with a working distance of 2 cm and a magnification of 600X has been built, and will be used for a study of phase transformations in steel at elevated temperatures. Studies of the gas chromizing process indicate that chromium is being deposited from an ionized gas rather than by a simple replacement reaction. Other investigations that might be mentioned include continuous casting, fatigue and the sigma phase in alloy steels.

In the *corrosion laboratory* of U. R. Evans, considerable work is being done on oxidation, passivity and stress corrosion.

(2) In the Cavendish Laboratory, the *metal physics* group under E. Orowan is concentrating on deformation studies. The creep of zinc and cadmium single crystals is being investigated with the particular object of determining whether or not viscous (i.e. non-transient creep) can occur in single crystals, and a machine has been constructed for the study of transient creep in polycrystalline wires. Previous work on the deformation of transparent crystals such as AgCl has been extended to include a study of the creep properties of single crystals of NaCl. The problems of yielding and aging in mild steel, deformation bands in aluminum, martensite formation and the mechanism of rapid plastic deformation at high temperatures are also being investigated.

Also in the Cavendish Laboratory, the *x-ray crystallography* group under W. H. Taylor is working on some problems in metallurgy. By passage through lead glass capillary tubing, x-ray microbeams have been obtained as narrow as 1 micron although most of the work is done with beams about 35 microns in diameter. The high resolution afforded by the x-ray microbeam is being applied to problems such as line broadening, plastic deformation and the difference between material in and near grain boundaries. In collaboration with G. V. Raynor of Birmingham University, the structure of ternary aluminum-rich alloys containing transition elements is being investigated and electron density projections are being made to show the actual electron distribution.

At the H. W. Wills Physical Laboratory of the University of Bristol, a theoretical group under N. F. Mott is working on the application of dislocation theory to such problems as plastic deformation, recrystallization, crystal growth and formation of oriented overgrowths. Some experimental work is being done on creep.

Phase equilibria continues to be the main field of investigation in the laboratory of William Hume-Rothery in the Inorganic Chemistry Department of Oxford University; the determination of the binary phase diagrams of several transition elements is currently receiving the most attention. The effect of thermal cycling on precipitation in alpha-beta brass is being investigated and a general survey of gamma phases in binary alloy systems is being made.

Among other things, the metallography of titanium and the properties of pure binary fer-

rites are being studied at the Metallurgy Division of the National Physical Laboratory under the direction of N. P. Allen. The purpose of this latter work is to determine the fracture characteristics of pure ferrites as a function of temperature; the first hurdle in this investigation has been successfully passed, namely the development of a method of regularly making 25-lb ingots of iron containing less than 0.01 pct of any one element.

**France:** Most basic research in physical metallurgy in France is centered in Paris, and in laboratories supported by government or industry. If any one subject must be singled out as typical of French investigation, that subject would probably be the fine structure of crystals, both in itself and as a variable in various metallurgical transformations.

At the Conservatoire National des Arts et Metiers, a group under A. Guinier is applying x-ray diffraction to the study of the mechanism of precipitation from solid solution and the relative perfection of grains during cold work and recrystallization. A focussing Laue method of high precision has been developed which shows, by the fine structure of one Laue spot, the degree of non-parallelism existing among the diffracting planes. A diffuse spot from a strained specimen gives evidence of lattice curvature: low temperature annealing of the same specimen causes the lattice to break up into a set of nearly perfect blocks having slightly different orientation (polygonization) as evidenced by the striated appearance of the Laue spot.

The Centre de Recherches at the Ecole des Mines is supported by IRSID (Institut de Recherches de la Siderurgie), a cooperative organization of steel manufacturers and the government. Under the direction of C. Crussard, investigations of precipitation, recrystallization, creep and transformations in steel are being made. Much use is made of thermoelectric power as a means of following changes in composition and relative perfection of crystals. A back reflection x-ray method of measuring the lattice parameters of individual grains has been developed and it is found that both the average parameter and the spread in parameter decrease during grain growth.

At the Vitry laboratory, just outside Paris, of the Centre National de la Recherche Scientifique, metallurgical research is conducted under the general direction of G. Chaudron. Gas-metal systems are being investigated as well as the mechanism of metallic oxidation. Studies of the sub-grain structure of recrystallized aluminum (which has been identified with the grain polygonization referred to above) are being continued under the direction of P. Lacombe and experiments on age-hardening aluminum alloys are being interpreted in terms of the sub-grain structure observed.

**Belgium:** A large part of the metallurgical research being done in Belgium is supported cooperatively by government and industry and administered by IRSIA (Institut pour l'Encouragement de la Recherche Scientifique dans l'Industrie et l'Agriculture). At Liege, an IRSIA-supported laboratory directed by P. Coheur is doing some work in physical metallurgy, besides a



rather larger amount on the process side. Plastic deformation is being investigated by means of magnetic measurements on stressed specimens and the behaviour of Cr-Mo steel during creep tests is being followed by electron microscopy.

At Charlerio, there is another IRSIA-supported laboratory, directed by J. Massinon, where work is being done on the fracture of metals. Direct microscopic examination of zinc during fracture has led to the conclusion that brittle fracture (i.e. pure cleavage right across a grain) is more common than is ordinarily supposed and occurs as a part of many "normal" fractures.

At the Ecole Polytechnique on Mons, G. A. Homes is directing work on recrystallization and on the structure of electrodeposited metal.

**Holland:** While there is no department of physical metallurgy as such in Dutch universities, research in physical metallurgy is conducted in two departments of the Technical University of Delft:

(1) The Department of Physical Chemistry. Here W. G. Burgers is continuing his investigations of recrystallization and grain growth phenomena, chiefly from the viewpoint of orientation relationships.

(2) The Department of Technical Physics. Under the direction of M. J. Druyvestyn, the inter-relation of stress, strain and electrical resistivity are being investigated in the plastic region.

**Sweden:** At the Royal Institute of Technology in Stockholm, several research projects are active in the Division of Metallography, directed by Axel Hultgren. (The term "metallography" in Sweden has the larger significance of "physical metallurgy" as used in the United States.) Chief among these is a study of the isothermal decomposition of austenite; the effect of alloying elements on the formation of pearlite and the partition of these elements between the various carbides is being investigated. Former work on the plastic deformation of steel at elevated temperatures has been extended to non-ferrous metals. G. Borelius, head of the Division of Physics, is continuing his work on the kinetics of solid state reaction. The crystallization of selenium, order-disorder transformations and precipitation from solid solutions are being studied by various means, including x-ray diffraction, calorimetry and measurement of electrical resistance.

The Institute for Metal Research in Stockholm, directed by Erik Rudberg, is supported by industry and government. Strain aging of mild steel and constitution of aluminum-rich alloys are typical of the problems being investigated.

Work on the crystal structure of intermediate phases and solid solutions is continuing at the University of Upsala under the direction of Gunnar Hägg, head of the Institute of Chemistry. A large amount of work has been done on the oxides and borides of the transition elements.

**Norway:** The only place in Norway where metallurgy is taught, and research conducted, on a university level is at the Norwegian Technical University in Trondheim. Very little research is being done at the present time 'due to postwar difficulties in obtaining sufficient staff and equip-

ment. A. B. Winterbottom, of the Metallurgy Department, is studying the formation of thin films on metals by means of the polarization spectrometer.

**Germany:** The various Kaiser Wilhelm research institutes of prewar years have been renamed in honor of Max Planck, and the most active centers of metallurgical research in Germany today are the Max Planck Institut für Eisenforschung in Dusseldorf (British Zone) and the Max Planck Institut für Metallforschung in Stuttgart (U.S. Zone).

The Institut für Eisenforschung, directed by Fritz Wever, functions today with most of its prewar equipment intact. Transformations in steel and the effects of inclusions on properties are being investigated, as well as deformation under short time stresses produced by shock waves.

The Institut für Metallforschung is under the direction of Werner Köster and its work is confined largely to nonferrous metallurgy. Very little equipment survived the war so that this laboratory is working on a rather reduced scale when compared with the prewar institute. Problems such as age hardening, intermediate phase crystal structure and phase equilibria are being investigated. There is a considerable interplay of staff and research projects between this institute and two other Stuttgart laboratories, namely the Technische Hochschule and the Roentgen Institut. U. Dehlinger and A. Kochendorfer at the Hochschule are doing theoretical work on plastic deformation while R. Glocker, director of the Roentgen Institute, is continuing his x-ray diffraction studies on such subjects as stresses in metals, the amorphous state in such elements as Se, As, Sb and Te, and the structure of liquids.

**Other Countries:** The writer has no personal knowledge of the status of metallurgical research in Denmark, Switzerland, Austria or Spain but it is his impression that the amount of fundamental work done in these countries is not large. In Italy, although considerable applied research is being carried out by industrial laboratories, there is at the present time very little basic work being done in physical metallurgy, according to information received by the writer.

**General:** There is a definite impression that proportionally more physicists are engaged in metallurgical research in Europe than in America, this being particularly true in England. Dislocation theory is popular and widely used in attempts to solve metallurgical problems. In their work on metals, European investigators are always extremely conscious of the fact that any actual metal lattice does not correspond to an illustration in a crystallography textbook but is full of various kinds of faults and imperfections which profoundly influence its properties and behavior.

In the field of x-ray diffraction, the large amount of more or less homemade tubes with special characteristics is noteworthy. Tubes with rotating anodes and electrostatically focussed electron beams are not uncommon and these furnish the investigators with an almost point source of high intensity x-rays. In some experiments, such as Guinier's focussing Laue method, such a source is not a luxury but a necessity.



# Outstanding Blast Furnace Operation

by Kurt Neustaetter

**D**URING the last few years a number of reports were given before the Blast Furnace, Coke Oven and Raw Materials committee describing blast furnace operation successful far beyond the performance customary in the industry. Reference is being made particularly to cases in which due to some change in operating methods an increase in daily tonnage of hundreds of tons was accomplished. Naturally management and blast furnace operators throughout the industry consider these cases to be models, and the operators are eager to show that they can do as well as the next fellow.

To duplicate blast furnace performance has always been a hard job. Only rarely has it been done even under allegedly equal conditions. Not only is it hard to imitate others, but quite often an operator cannot even repeat his own feats.

In order to profit from another operator's experience it is necessary to study the conditions under which his success was accomplished. It is not sufficient to concentrate on any one new feature which he holds responsible for the good performance, instead thorough attention must be given to all measurable circumstances.

Consider first the excellent performances obtained by Hoffman, Dobscha and Saussaman as well as some figures concerning high top pressure. In the first three cases top results were ascribed to raw materials and in the latter case a different operating method is dealt with.

---

*Kurt Neustaetter is Blast Furnace Engineer, Inland Steel Co., East Chicago, Ind.*

*This paper will be presented before the Annual Blast Furnace, Coke Oven and Raw Materials Conference, April 12, Cincinnati.*

---

Last year during the discussion of the relative merits of large and small blast furnaces C. F. Hoffman<sup>1</sup> of Sparrows Point showed figures concerning a 28-ft furnace producing an average of 1753 tons per day and a 19-ft furnace producing

an average of 893 tons per day. Nobody will deny that such performances are outstanding.

Sparrows Point uses a large amount of imported ore of an iron content of approximately 58 pct, and of a silica content of 9 pct. At first glance every operator will say: "Well, if you give me an ore with 58 pct iron I can do just as well". It is necessary, however, to consider that the ores in question contain less than 1 pct moisture, while the average Mesabi ore contains perhaps 12 pct moisture. Figured on a dry basis, the standard Mesabi ore contains approximately 55 to 56 pct iron, and approximately 10 pct silica. This gives the ores used at Sparrows Point an advantage in slag volume, but Mr. Hoffman has said that their slag volume during the time in question was 979 lb, which is not overwhelmingly low. According to the rule that a 10 pct decrease in slag volume will bring about a 4 pct increase in production and assuming a slag volume of 1100 lb for Mesabi practice Mr. Hoffman's success cannot be explained by slag volume alone. This is also borne out by the fact that his coke consumptions of 1628 and 1642 lb per ton of iron respectively are not unusual for good Mesabi practice, and his coke certainly is not better than average. Unusual, however, is his high blast heat of 1440°F and this points toward a loose stock column caused by good ore structure. With Mesabi ore it would be impossible to use such blast heats and at the same time to maintain full wind at coke rates as low as those just mentioned. The wind rates of 90,800 cfm on the large furnace and 50,200 cfm on the small furnace are comparable with Mesabi practice at least for the small furnace. It may be concluded that aside from skillful handling of the furnaces at least two factors inherent in the raw materials are responsible for the excellent results achieved at Sparrows Point: namely slag volume and structure.

Next, consider H. F. Dobscha's<sup>2,3</sup> experiments using sinter and nodules.

In his 1946 paper before the Iron and Steel Institute, Mr. Dobscha describes a program in-



volving two 27-ft 6-in. furnaces. A mixture of beneficiated ores (coarse ores, nodules and concentrates) was charged on one furnace while normal ore was charged on the other furnace for alternating two-month periods; coke and flux were of the same quality in both cases. When using the beneficiated burdens the furnaces produced 1591 tons per day as against 1400 tons with the normal burden. At the same time the rates of coke consumption were 1670 and 1816 lb of coke per ton of iron produced. Slag volumes were 954 lb per ton of iron for the beneficiated ores, 1100 lb per ton of iron for the normal ore burden. The wind rate was 2.1 pct higher with the beneficiated burden. The drop in slag volume was 13.3 pct, pointing to a production increase of  $13.3 \times 0.4 = 5.3$  pct. The actual increase in production, however, was 11.4 pct. Only 7.4 pct are accounted for by the above two items. Flue dust production was 22 lb per ton of iron lower in the case of the beneficiated ore. At 50 pct Fe in the flue dust this accounts for 0.5 pct increase in production. This leaves 3.5 pct for better reduction due to structure and unaccounted for. Production increases due to improved slag volume and due to better reduction should be accompanied by promotional decreases in coke consumption. Production increases due to higher wind or decreased flue dust production should not be accompanied by a decrease in coke consumption. Adding 5.3 pct for slag volume and 3.5 for greater efficiency, the result is 8.8 pct, which is very close to the 8 pct improvement in coke consumption accomplished by Mr. Dobscha; 0.8 pct of the improvement remain unexplained, especially since the blast heat was actually lower in the case of the beneficiated ore. The results just shown suggest that the combined effects of two favorable factors, in this case slag volume and structure, might add up to results more favorable than the sum of the effects of the factors when taken separately.

**During the 1948 AIME meeting** Mr. Dobscha showed what happened when the nodules were replaced with practically the same amount of sinter, while a similar amount of coarse ore and concentrates were used as in the previous test. The improvement in performance was still more amazing than in the first test. The iron production rose from 1324 tons per day to 1605 tons per day, or 21.1 pct; coke consumption dropped from 1859 lb to 1510 lb, or 15.3 pct. The rise in wind was 2.4 pct. Slag volume dropped from 1264 lb to 968 lb, or 23.4 pct; and thus the anticipated increase in tonnage from slag volume alone would be  $23.4 \times 0.4 = 9.4$  pct. Decrease in flue dust was 57 lb per ton of iron at 50 pct Fe, accounting for a production increase of 1.4 pct. Again the blast heat for the beneficiated burden was lower. The increase in tonnage of 21.1 pct can be split into the following items: slag volume: 9.4 pct, wind: 2.4 pct, flue dust: 1.4 pct, or a total of 13.2 pct. This leaves 7.9 pct for better reduction and factors not accounted for. Coke consumption was reduced by 15.3 pct; better reduction and slag volume add up to 17.3 pct. Again, as in the case of the nodules, the improvement in coke consumption is slightly less than the increase in production due to factors affecting both production and coke consumption. While in the first case 0.8 pct were unaccounted for, there is 2 pct un-

accounted for in this case. This strengthens the impression that the total improvement may differ from the sum of the effects of slag volume, increased wind, less flue dust and better reduction. This phenomenon is strongly reminiscent of eutectics as experienced in alloys.

Another case of a favorable combination of slag volume and structure was described during the 1948 AIME meeting by J. D. Saussaman.<sup>4</sup> Unfortunately no operating figures are given for operating results before the installation of an elaborate ore bedding and treating system. A tonnage of 1259 tons on a 25 ft 6 in. furnace with a 1416 lb coke consumption and a slag volume of 837 lb per ton of iron are hard to match when the lack of uniformity in Western ores is considered, as described in E. J. Duffy's<sup>5</sup> paper on the ore preparation for the Kaiser furnaces.

Consider next an entirely different method of achieving greatly improved tonnages: namely high top pressure. As an example the author cites a set of figures communicated through the courtesy of Arthur D. Little, Inc., concerning two furnaces of the Republic Steel Corp. in Youngstown. The figures given cover Nos. 1 and 3 furnaces for the period October 1946 to September 1949. No. 1 furnace operated under normal top pressure, No. 3 furnace operated under top pressures from 4 to 12 psi, predominantly in the range of 9-12 psi. The iron content of the charge was the same for both furnaces, while the furnace with normal top pressure was slightly favored in as far as coke quality was concerned. The average tonnage on No. 1 furnace for the three-year period was 959 tons, the average coke consumption was 1981 lb per ton of iron. For No. 3 furnace the figures were 1103 tons of iron and 1790 lb of coke per ton of iron. Although these data indicate considerably improved performance with high top pressure, the tonnage and coke on the high-pressure furnace are not outstanding for a furnace of that size (25 ft 6 in. hearth diameter). It would be interesting to see what would happen if beneficiated materials were combined with high top pressure.

Another instance of one favorable factor not producing outstanding results is Inland's experience with the use of sinter. Many inconclusive studies were made using varying percentages of sinter. A beneficial or harmful influence just could not be established. A report submitted about the use of sinter on No. 5 furnace for the year of 1942 is typical. The amount of sinter per charge used on No. 5 blast furnace varied as follows:

Date	Lb of Sinter per Charge	Daily Tonnage	Days
Jan. 1-6	5000	1211	6
Jan. 7-11	6000	1192	5
Jan. 12-19	5000	1214	8
Jan. 20-Feb. 9	4000	1248	21
Feb. 10-March 13	5000	1260	32
March 14-30	4000	1279	17
March 31-Aug. 28	6000	1265	151
Aug. 29-Oct. 9	4500	1210	42
Oct. 10-Nov. 19			
(minus 2 days)	6000	1212	39
Nov. 20-December	4000	1146	42
<b>TOTAL</b>			<b>363</b>



For the different amounts of sinter used, there are the following averages:

Lb of Sinter per Charge	Daily Tonnage	Days
4000	1201	80
4500	1210	42
5000	1213	14
6000	1253	195

At first glance this looks like an excellent correlation, but when examining the details there appear grave doubts about its validity. During the first half of the year there was a gradual increase in tonnage independent of the amount of sinter used, while a general drop in production occurred during the second half of the year. It might also be added that it would be strange if there would be a small increase of tonnage when raising the sinter from 4000 to 5000 lb, and a big jump when going from 5000 to 6000 lb. Furthermore, the highest tonnage produced was during a 17-day period when 4000 lb of sinter were used.

Having come to the conclusion that sinter will not do much good, there is need to re-examine the Dobscha and Saussaman papers which claim great improvements. Seeing, however, that in both cases low slag volumes were employed, it is thought that their success might be due not to sinter but to slag volume. Inland sinter is produced from flue dust, filter cake and the fines of screened standard Mesabi ore. In Mesabi ores the finer particles usually are higher in silica and lower in iron; therefore, when replacing standard ores in the burden with the equivalent amount of sinter it is actually found that there is a slight increase in slag volume.

In order to decide this question, a test was conducted at Inland from January to April, 1949. All low-slag-volume ores were assembled at No. 6 furnace. This furnace has a hearth diameter of 25 ft 9 in. Operating results follow:

	Daily Ton- nage	Coke Con- sump- tion Lb. Per Ton of Iron	Slag Vol- ume, Lb. Per Ton of Iron	Pig/Coke Ratio	Wind CFM
December, 1948	1141	1798	1036	1.25	66,984
January, 1949	1154	1713	842	1.33	64,104
February, 1949	1151	1734	935	1.32	64,215
March, 1949	1182	1784	1028	1.25	69,975
April, 1949	1220	1742	1020	1.23	69,678
May, 1949	1170	1743	1076	1.24	70,467

During January the increase in tonnage remained far behind what was expected. This was not due to failure to take burden, but to failure to take wind. Although sufficient coke was available, almost 3000 cf less wind was blown during January than there was during December. The decrease in wind was 4.3 pct, the decrease in slag volume was 18.7 pct. According to the rule that a 10 pct decrease in slag volume should bring a 4 pct increase in production—provided everything else remains equal—there should have been an increase in tonnage of 7.48 pct, or a tonnage of 1226 tons. Actually, in spite of a burden increase of 6.4 pct, the tonnage increased only 1.1 pct.

Searching for a reason for the inadequate improvement of operation, prevailing conditions were compared with previous ones when slag vol-

umes as low as 842 were experienced. It was found that this was a time of most successful operation and that coke stability was over 50 pct while in January, 1949, it was only 44.2 pct. This ties in very well with the fact that the furnace did not take the wind. It indicated that the coke was just not good enough, i.e., it did not provide sufficient voids for low-slag-volume operation.

During February the slag volume was raised from 842 to 935 lb per ton of iron. Nevertheless, the tonnage was only 3 tons below January's. At the same time the coke consumption increased 21 lb. The average burden carried dropped from 1.33 to 1.32. Practically the same amount of wind was blown; the furnace did not take the wind in either month.

On March 1, the slag volume was raised again, this time from 935 lb per ton of iron to 1028 lb per ton of iron. The difficulties in blowing wind disappeared but there was also a decided increase in coke consumption and a decided drop in burden carried. Even better operation was achieved in April with approximately the same slag volume. During May, when the slag volume was raised to 1076 lb per ton of iron, there was a setback.

In summarizing this test it may be said that the attempt at imitation was a complete failure. In this connection the fact could also be cited that it is not possible to profit from a cut in coke consumption unless replacement of the voids deprived to the furnace by this very cut. This subject has been treated in detail in two previous papers.<sup>6,7</sup>

It was learned from this test that low slag volume alone will not procure beneficial results just as little as sinter alone will do so. Good structure seems to require low slag volume. Low slag volume seems to require good ore structure. Just one favorable factor, even if it is unusually favorable, will not produce record tonnage. The absence of unfavorable factors, and especially of raw material variations, seems to be at least as important.

Three examples could be cited where one unfavorable factor completely upset conditions.

J. F. Peters<sup>8</sup> paper on the correlation of coke braze production and furnace operation showed that under the conditions of plant No. 3 at Inland an increase of coke braze produced while handling the coke has catastrophic consequences for furnace operation. The unfavorable factor of the braze more than offsets a decrease in slag volume, as shown in the following example:

	Nov. 1-15, 1947	Nov. 16-30, 1947
Average per day, tons	1060	966
Coke per ton of pig, lb	1818	1894
Wind per min, cf	70,500	68,400
Coke ash, pct	10.78	10.33
Braze, pct	8.8	10.1

The coke braze produced outside the furnace is a good indicator of the amount which will be produced inside the furnace. This is clearly shown by reduced wind with increased braze.

That the disastrous effect of poor coke—usually a consequence of poor coal—cannot be overcome by some other highly favorable factor can easily be seen when the detailed data on high pressure operation at Youngstown Nos. 1 and 3



are examined. Due to a coal strike inferior coal had to be used during April and part of May, 1948. In the following table the data for production and coke consumption are given for the periods from January to March and for April, May and June to August, 1948.

1948	No. 1 Furnace		No. 3 Furnace	
	Tonnage	Coke Consumption	Tonnage	Coke Consumption
January-March	961	2006	1105	1745
April*	665	2282	919	2014
May	840	2031	951	1786
June-August	915	2104	1106	1780

\* Two weeks operation only.

Both furnaces were affected about equally and the top pressure did not save the day.

The importance of coal quality for successful furnace operation also was stressed in E. J. Gardner's<sup>9</sup> paper on coal preparation for a 2000-ton blast furnace.

In order not to appear biased against coke the author would like to cite a case where irregular ore analysis was extremely harmful.

During November, 1948, due to mine breakdowns and freezing conditions, Inland received some off-grade ore. One grade that had silica contents varying between 9.43 and 9.80 pct during October, showed silica contents of 11.17, 10.94, 9.34 and 10.02 pct during November. Another grade the silica content of which had varied between 9.00 and 9.69 pct changed to silica contents of 10.71, 10.34, 11.56, 11.00 and 9.47 pct. For the two furnaces at plant No. 3 the off-casts due to high sulphur increased 200 pct and the tonnage dropped 96 tons per day.

It was not so much the higher silica content that hurt but the variations in silica content. The amounts of stone required varied so rapidly that it became impossible to keep up with the ever changing requirements. The amounts of the two grades used on the furnaces were so large that stone requirements varied up to 1200 lb per charge for the different analyses of the several cargoes. The limestone charge is based on the assumption that the cargoes are uniform; actually they vary from hatch to hatch and there is no telling just how large the difference in stone requirement is from one extreme to another.

The conditions under which each Inland furnace achieved its record production are as follows:

**No. 1 Furnace:** 20-ft hearth diameter. The furnace produced 1013 tons per day during the month of November 1943. The coke consumption was 1515 lb per ton of iron; 165 tons of scrap in the form of borings and turnings were used per day, decreasing the slag volume to 777 lb per ton of iron. The amount of Marquette ore used was somewhat higher than usual, coke stability was 51.6 pct, the average wind was 56284 cfm. This is an obvious case of low slag volume combined with good physical quality of the raw materials.

**No. 2 Furnace:** 19-ft hearth diameter. This furnace is too low for its hearth diameter and has always required a high coke rate. The furnace does not use scrap as explained in detail in the paper on the use of scrap in the blast furnace<sup>7</sup> in 1946. The furnace produced 913 tons with a coke

consumption of 1715 lb per ton of iron, the wind rate was 53,735 cfm, the slag volume was 841 lb per ton of iron. Coke stability was 51.1 pct. Conditions were similar to those prevailing at No. 1 furnace.

**No. 3 Furnace:** 17-ft 3-in. hearth diameter. During October, 1942, the furnace produced 844 tons per day with a coke consumption of 1549 lb per ton of iron. 78 tons of scrap were used per day. Slag volume was 809 lb per ton of iron, wind blown 47,494 cfm, coke stability 52.5 pct.

**No. 4 Furnace** followed a different pattern and will be discussed later.

**No. 5 Furnace:** 25-ft hearth diameter\*. 1309 tons per day were produced during July, 1943, with a coke consumption of 1498 lb per ton of iron, 113 tons of scrap per day, and a slag volume of 746 lb per ton of iron. Wind blown was 67,494 cfm; coke stability 50.9 pct.

**No. 6 Furnace:** 25-ft hearth diameter. (This furnace was since enlarged to 25 ft 9 in.) Production was 1279 tons per day during October, 1943, coke consumption was 1473 lb per ton of iron, the amount of scrap charged was 95 tons per day, slag volume was 829 lb per ton of iron. Wind blown was 66,844 cfm; coke stability 51.1 pct.

All these records follow exactly the same pattern, and they were achieved within the short period of roughly one year. The same low slag volume burden and high stability coke were used in all cases. At the time No. 4 furnace had a very old lining and the furnaces at plant No. 3 were not yet in operation. After 1943 coke quality deteriorated rapidly, the coke ash started to rise and so did the silica content of the ores.

Production records on the following furnaces were set at times of temporarily improved coke conditions.

**"B" Furnace:** 25-ft 9-in. hearth diameter. Production of 1265 tons per day was achieved with a coke consumption of 1602 lb in November, 1944, during extensive tests comparing the value of several types of coal for blast furnace coke. These tests have been described by Mr. Gardner at several meetings and the results have appeared in book form.<sup>10</sup>

**"A" Furnace:** 25-ft 9-in. hearth diameter. The furnace produced 1173 tons per day during June, 1949, with 1715 lb of coke per ton of iron, no scrap, a slag volume of 1073 lb per ton of iron, 73,178 cf of wind per min and a coke stability of 54.4 pct. Thus coke stability was as good as in the plant No. 2 cases cited, but slag volume was relatively high; this furnace which was started in 1947 never has had a chance to compete with the other large furnaces.

**No. 4 Furnace:** 20-ft hearth diameter. During May 1945, 1021 tons per day were produced with 1563 lb of coke per ton of iron, 106 tons of scrap per day, a slag volume of 914 lb per ton of iron and 58,511 cf of wind per min. Two grades of coke were used, 60 pct of plant No. 3 coke containing 10.40 pct ash and having 40.6 pct stability and an average size of 2.62 in. and 40 pct of plant No. 2 coke containing 7.51 pct ash, having 43.6 pct stability and an average size of 2.14 in; 14 pct screened ore and 24 pct sinter were used in the burden.

Here is a case of excellent structure of the iron-bearing material and of a slag volume which



though higher in most other cases was still relatively low. The coke was quite poor. The mixture of the two types of coke seems to be of particular interest. The variations in coke ash are equalized. It is also believed that there is a strong probability that the larger-sized, high-ash coke and the smaller-sized low-ash coke supplemented each other and formed a most favorable "eutectic" mixture having the good features of both cokes while pushing the less desirable ones into the background. Mention should be made that the previous record on this furnace was 979 tons during February, 1940, and that even the latter figure has not been approached since. During the month that followed, June, 1945, the coke mix could not be maintained. The furnace produced 914 tons of plant No. 3 coke from June 9-23 and 963 tons of plant No. 2 coke from June 24-July 31.

This may be the occasion to put in a plug for coke burdening which should be done with just as much care and thought as is generally bestowed on ore burdening.

**Summary:** Cases of top production of blast furnaces were compared and it was found that only one favorable condition such as low slag volume, good structure of ore or coke or high top pressure will not result in record production. A number of favorable factors combined with the absence of unfavorable factors are necessary to achieve best results. On the other hand, it is not necessary to

have a number of unfavorable factors in order to achieve poor results. One unfavorable factor is entirely sufficient.

### References

- <sup>1</sup>C. F. Hoffman, Operating Differences between Large and Small Furnaces, Trans. AIME 1949, p. 121/2.
- <sup>2</sup>H. F. Dobscha, Effect of Sized and Nodulized Mesabi Iron Ores on Blast Furnace Performance, Blast Furnace & Steel Plant 1946, p. 979/985.
- <sup>3</sup>H. F. Dobscha, Effect of Sized and Sintered Mesabi Iron Ores on Blast Furnace Performance, Trans. AIME 1948, p. 49/67.
- <sup>4</sup>J. D. Saussaman, Sintering Practice at Fontana, Cal., Trans. AIME 1948, p. 95/106.
- <sup>5</sup>E. J. Duffy, Ore Blending, Trans. AIME 1949, p. 204/210.
- <sup>6</sup>K. Neustaetter, Observations on Gas Flow and Coke Consumption, Blast Furnace & Steel Plant 1945, p. 825-829.
- <sup>7</sup>K. Neustaetter, Use of Scrap in the Blast Furnace, Blast Furnace & Steel Plant 1946, p. 592-599, 627-632.
- <sup>8</sup>J. F. Peters, Effect of the Coke Stability on the Operation of the Blast Furnace, Trans. AIME 1949, p. 7-31.
- <sup>9</sup>E. J. Gardner, Coal Preparation for the Production of Coke to be Used in a 2000 Ton Blast Furnace, Blast Furnace & Steel Plant 1948, p. 1463/1464.
- <sup>10</sup>N. I. Isenberg, H. W. Jackman and E. J. Gardner: Investigation of Beckley Seam, Inland Steel Co., July 1945.

## Uranium in Rubber Research

THE search for peacetime applications of base materials used in the atomic bomb and developments arising from atomic research goes incessantly onward, with each day bringing to light new and sometimes unusual applications of these materials and developments. The use of radioactive isotopes has spread into many fields, such as metallurgy, medicine, oil refining, textile and rubber.

Recently the B. F. Goodrich Co. received shipment of  $\frac{1}{4}$  lb of uranium from the U. S. Government, for use at the Goodrich Research Center at Breckville, Ohio. This uranium will find use in electron microscope work by "shadow casting" microscopic samples showing low contrast.

Some objects studied with the electron microscope in the rubber industry are so minute that even with the electron microscope difficulty is encountered in examining them and, particularly, photographing them. Uranium allows such particles to be photographed more distinctly.

The uranium is being held by an employee of the Goodrich Research Center.



*Attention IMD authors. See announcement on p. 622*



# Superimposed Precipitators

by R. E. Touzalin

**T**HE theory of operation and details of design of equipment for the fine cleaning of blast-furnace gas in a tower scrubber and electrostatic precipitator combination have been discussed in a number of excellent papers read in this country and abroad, during the past 12 yr. A further study of technical details at this time would not be of as much general interest as a brief description of several installations of this type of cleaning unit.

The primary gas washer with superimposed precipitator was originally developed as a means of conserving ground space at a Buffalo district plant where other types of gas-cleaning equipment could not be conveniently accommodated in the available space. Two 15-ft tower scrubbers, already on the site, were modernized, and on top of these were erected two Research Corp. electrical precipitator units, each comprising 238 8 in.-diam tubes, 15 ft long. The two cleaners have a rated capacity of 110,000 cfm of blast furnace gas, and during the early years of their service, were normally required to clean the gas production of two furnaces, usually amounting to about 125,000 cfm. The units have now been in service for over 12 yr, and have operated for a period as long as 27 months without shutdown. During that period, tests showed that outlet dust loading average about 0.01 grains per cu ft.

---

*R. E. Touzalin is an engineer, Arthur G. McKee & Co., Cleveland. This is a paper to be presented before the Annual Blast Furnace, Coke Oven and Raw Materials Conference, April 10-12, Cincinnati.*

---

furnaces served by these cleaners was increased, and it was necessary that additional gas-cleaning capacity be added. A third superimposed precipitator was erected in the 21-ft diam shell of an abandoned stove. This unit consisted of a single-stage tower scrubber under a standard Research Corp. precipitator having 116 12 in.-diam tubes, 15 ft long. Rated capacity of the cleaner was originally 60,000 cfm but, in order to handle the entire gas production of one furnace operating at high top pressure, the rating was later increased by adding 36 12-in. tubes to the precipitator unit. Operating information on the third cleaner is not available at this time.

The two latest installations of superimposed precipitators have been engineered under more favorable circumstances than the ones just described. In each case the cleaners were incorporated in a new plant where they could be fitted into the gas main and piping layout to the best advantage of the cleaners and the blast-furnace plant. Each installation is composed of three superimposed precipitators arranged to clean the gas production of two blast furnaces. Flexibility of operation is provided by arranging the three cleaners to be supplied with gas from a common raw gas main and to discharge into a common clean gas main.

The first installation of a three-unit cleaning system was made in 1943, in a Chicago district plant. Each of the three cleaner units (fig. 1) combines an 18-ft-diam, single-stage, three-tile-bank tower scrubber with a precipitator unit having 168 12-in.-diam tubes, 15 ft long. Rated total capacity of the three cleaners is 235,000 cfm of blast-furnace gas. The physical layout of this

During the late war, the production of the two



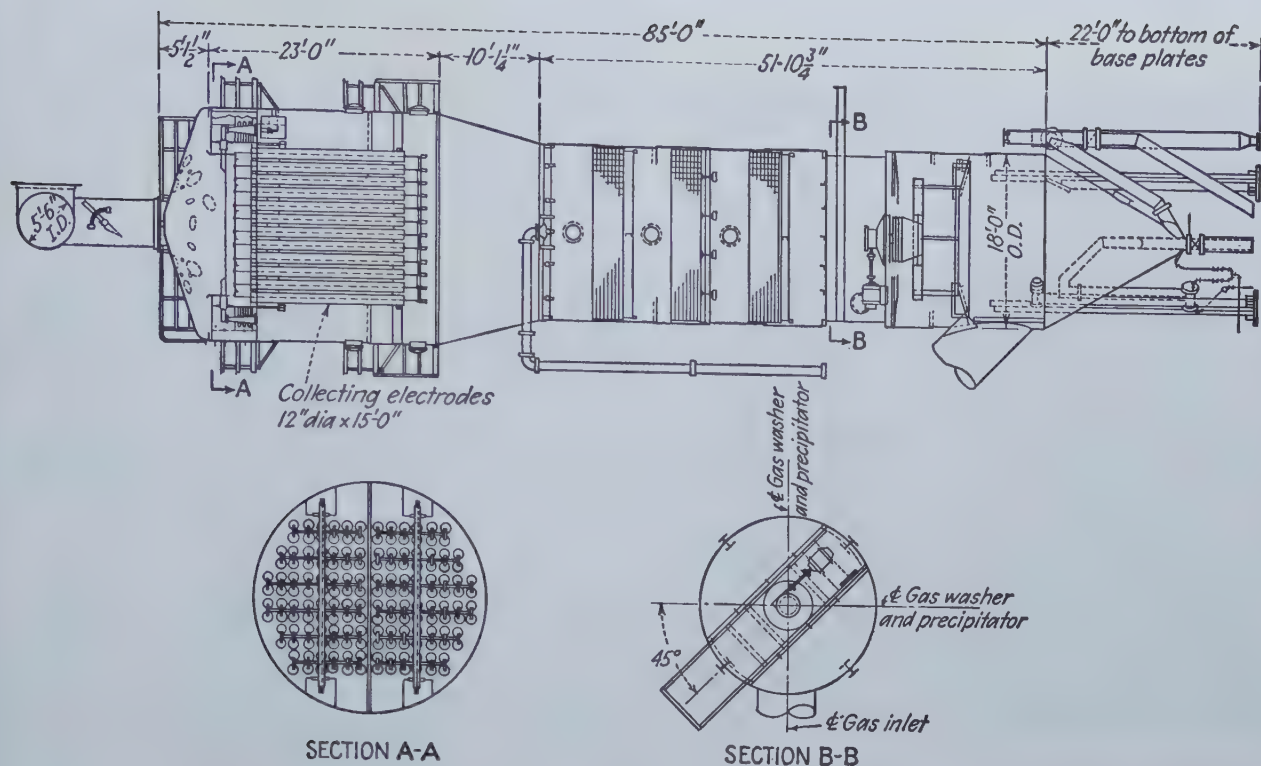


Fig. 1—McKee 18-ft one-stage gas washer with superimposed electrical precipitator designed by Research Corp.

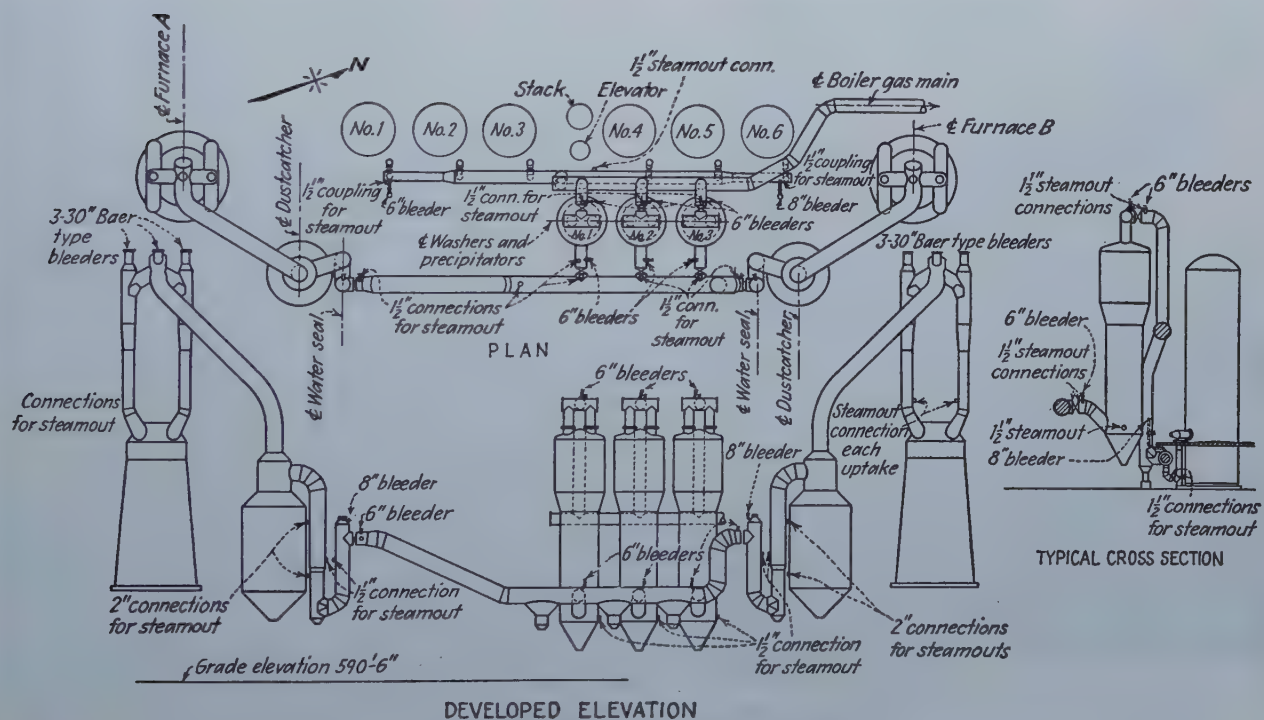


Fig. 2—General arrangement of gas mains shows the locations of steamout and bleeder connections.



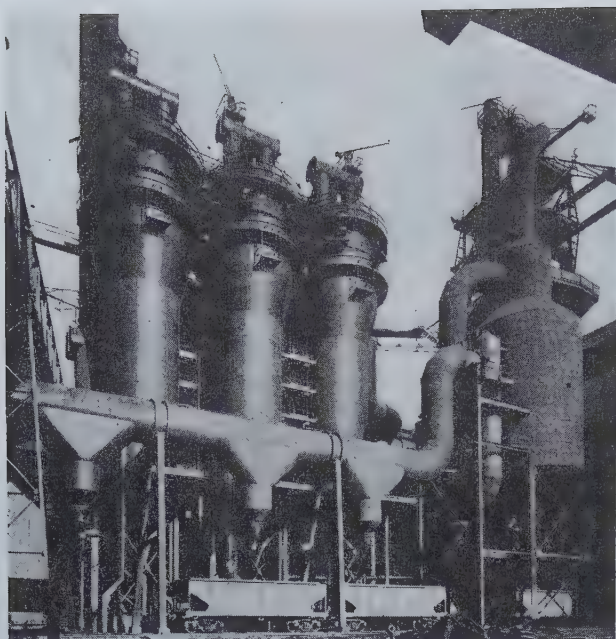


Fig. 3—General view of the gas-cleaning plant.



Fig. 4—Gas-cleaning units, showing the short gas mains and location relative to the stoves.

plant, shown in fig. 2, presents some interesting points regarding the arrangement of the cleaning units. The gas-cleaning plant shares the space between the two dust catchers with the pyrometer house and pump house. This places the cleaners near the stove platform and accessible from it at that level and from the stoves at the top level. (figs. 3 & 4) The location results in short gas mains and still allows ample room at the furnaces and stoves.

Table I shows the results of builder's tests of gas outlet cleanliness at various rates of gas flow through the cleaners. During six years of operation, outlet dust concentration tests taken, under all conditions, have averaged about 0.025 grains per cu ft.

The latest installation of a three-unit system, also in a Chicago district plant, was placed in operation in 1948. The layout of this plant presented

TABLE I

Gas Outlet Cleanliness of Different Rates of Gas Flow

Test No.	Cleaner No.	Gas Volume Per Cleaner, Cu Ft Per Min (60°F., 30" Hg)	Outlet Dust Concentration, Grains Per Cu Ft	Pct of Rated Capacity
1	2	51,200	0.0023	65
2	2	57,400	0.0038	73
3	3	79,200	0.0032	101
4	2	79,200	0.0192	101
5	2	79,200	0.0195	101
6	3	82,700	0.0079	106
7	3	82,700	0.0160	106
8	3	92,000	0.0277	117

a difficult problem, as an examination of figs. 5 and 6 will indicate. The distance between center-lines of the two 28-ft-hearth-diameter furnaces was 212 ft 4½ in. In the area between furnaces, there had to be located six stoves, two dustcatchers, water seals, a slag-granulating pit, pump house, sub station, pyrometer house, raw gas mains, clean gas mains and gas-cleaning plant. The use of superimposed precipitators was a "natural" for this application. Fig. 7 further illustrates the space limitations of the site. When this photograph was taken, the dustcatchers were not completely erected, and the raw-gas mains were conspicuously absent. In the center of the picture, directly under the inlets to the superimposed precipitators, may be seen the concrete wall which will form the pouring end of the granulating pit.

Fig. 8 is a cross-section through one of the cleaners of this system. The gas-washer units themselves are 21 ft in diam and 65 ft high, bendline to bendline. The washer is of the static-tower type, equipped with a lower bank 2 ft thick of 6 x 6-in. cross-partition rings, and three 3-ft banks of drip-point grid tile. There are two spray headers, one over the lower bank of rings and



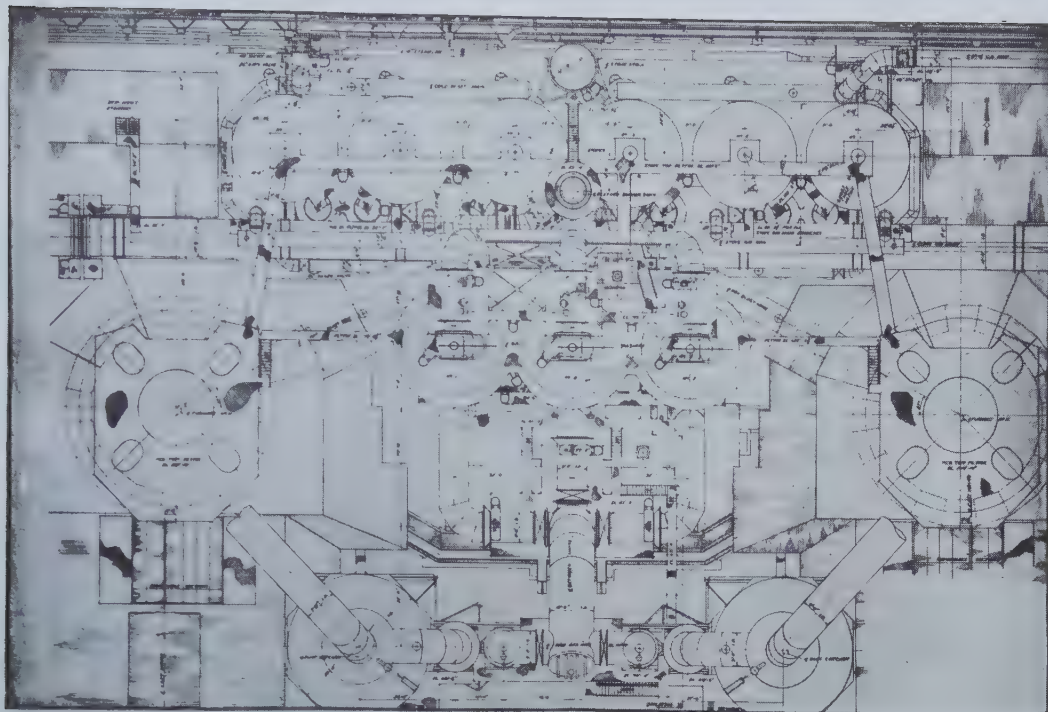


Fig. 5—General arrangement of stoves and gas-cleaning plants are shown in this top plan.

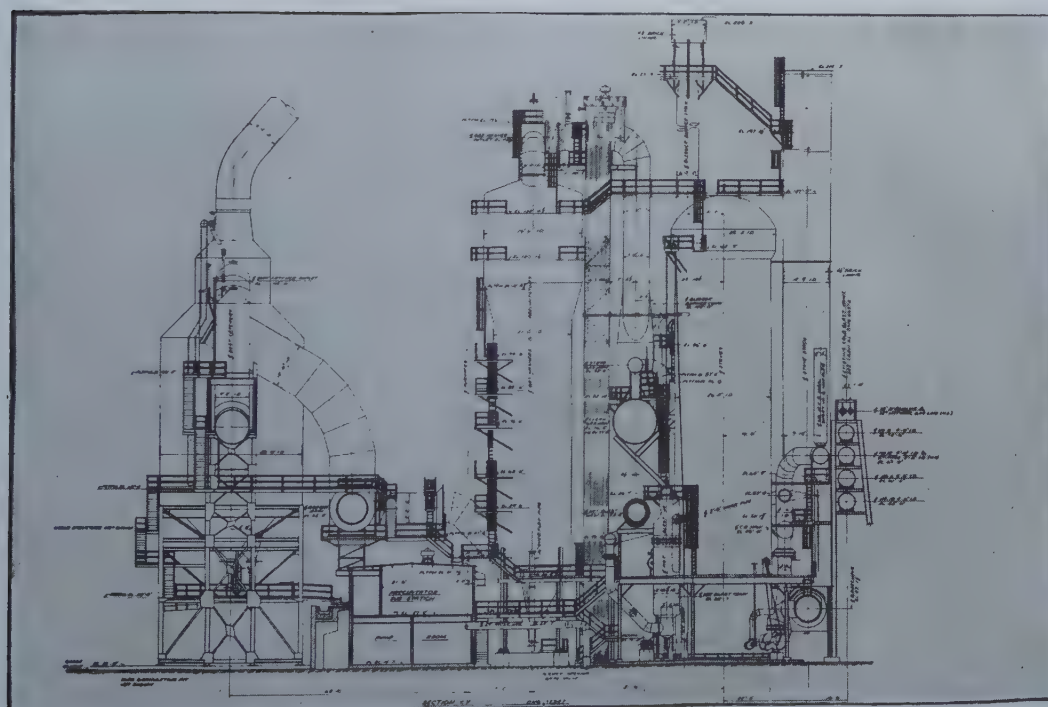


Fig. 6—General arrangement of stoves and gas-cleaning plant and blast furnaces Nos. 11 and 12.



one over the topmost bank of tile. Each header is equipped with 32 2-in. spray nozzles with 3/4-in. orifices. The 26 ft 6 in. diam precipitator unit is provided with 224 12-in. diam tubes, 15 ft long. The rated total capacity of the three cleaners is 300,000 cfm of blast-furnace gas.

Table II shows the results of builder's tests of gas outlet cleanliness at each of the three units. Flow of gas during these tests was 110 pct of the rated capacity.

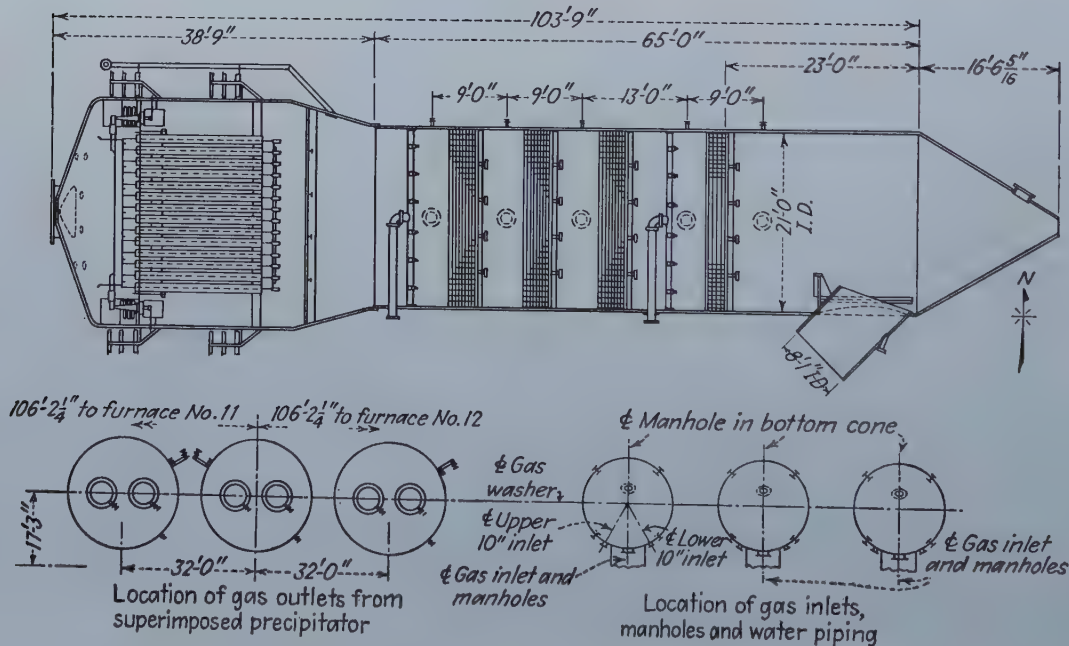
**TABLE II**  
**Gas Outlet Cleanliness—Builder's Tests**

Test No.	Cleaner No.	Gas Volume Per Cleaner, Cu Ft Per Min (60°F., 30" Hg)	Outlet Dust Concentration, Grains Per Cu Ft	Pct of Rated Capacity
1	1	110,000	0.0186	110
2	1	110,000	0.0167	110
3	1	110,000	0.0142	110
4	2	110,000	0.0250	110
5	2	110,000	0.0226	110
6	3	110,000	0.0202	110
7	3	110,000	0.0195	110

Space requirements in blast furnace plants have been, and will continue to be, an important factor in the selection of suitable fine gas-cleaning equipment. This factor is quite often as important in the design of a plant on a new site as it is in the enlargement of furnaces and auxiliaries on existing sites. In any such cases, the use of superimposed precipitators as a means of conserving ground space may be the practical solution of the problem.



**Fig. 7—Installation of a three-unit cleaning system in a Chicago-district plant.**



**Fig. 8—General arrangement of gas washer with superimposed precipitator.**



# Extractive Metallurgy of Aluminum

by R. S. Sherwin

The extractive metallurgy of primary aluminum from its ores is discussed with special attention to the production of alumina from high grade ores by the Bayer process, including differences between American and European practice and a brief description of some processes for lower grade ores and the electrolytic reduction of the oxide to aluminum.

METALLIC aluminum is not found in nature, but the oxides, hydroxides, and especially the silicates are plentiful. The estimated percentage of aluminum in the crust of the earth is about 8 pct while that of iron is about 5 pct. By far the larger portion of this is combined with silica in the form of various clay minerals and igneous silicate rocks. From the point of view of extractive metallurgy of aluminum, these are low grade ores while the better qualities of bauxite are the high grade ores. There have been various definitions of bauxite but perhaps the best definition at the present time is that bauxite is a rock or earth commonly used as an ore of aluminum or its salts in which the aluminum is present predominantly as a hydrate or a mixture of hydrates and hydrous oxides. It contains varying amounts of oxides of silicon, iron, and titanium and traces of compounds of some of the less common elements. The silica is mainly combined with alumina as clay or clay minerals which are hydrous aluminum silicates, although a part of it may be present as quartz sand. On the American continents, the alumina is mainly present as gibbsite,  $\text{Al}_2\text{O}_3 \cdot 3\text{H}_2\text{O}$ , and the same may be said of the best known deposits of the Dutch East Indies and some of the deposits in India. In France and other countries in Europe as well as in Africa, the alumina is present mainly as boehmite,  $\text{Al}_2\text{O}_3 \cdot \text{H}_2\text{O}$ , but in some of these deposits it is mixed with minor amounts of gibbsite. Some other deposits, such as those in the islands of Haiti and Jamaica, evidently contain two or more hydrates or hydrous oxides of alumina in varying proportions. Perhaps the main portion of the alumina may be present as gibbsite and boehmite with the proportion between the two varying rather widely.

In the silicate minerals, including clay, the alumina is chemically combined with silica and has not been separated satisfactorily by mechanical or physical ore-dressing methods. Low grade bauxites are mixtures of hydrates, usually gibbsite or boehmite, with clay, iron oxides, etc. In some low grade bauxites, it is possible to separate a portion of the gibbsite or boehmite, which may be present as relatively hard and coarse particles, from soft or finely divided clay minerals by log washing or similar methods. This

has been applied to the product of some mines or parts of them, but on other ores it is not applicable. In some cases the gibbsite or boehmite is almost as fine and soft and of nearly the same specific gravity as the clay minerals so that washing and gravity separations are not successful. The iron oxide, the clay minerals, and a part of the titanium minerals are often so finely dispersed in the ore that any of the physical mineral separation methods, including separations by gravity, particle size, flotation, and electrostatic or magnetic separation, have not been commercially successful except on relatively small lots of ore. For these reasons, the only available methods of separation on the general run of ores have been methods which would be classed as chemical rather than physical or mechanical separations.

Aluminum oxide can be reduced by carbon at temperatures above  $1800^\circ\text{C}$  to form metallic aluminum and aluminum carbide or nitride. The temperature for rapid reduction of aluminum oxide to metallic aluminum is about the boiling point of aluminum and above the temperatures necessary to reduce iron, silicon, and titanium so that the direct reduction of an aluminum ore with carbon will produce an alloy of aluminum, iron, titanium, silicon, etc., which may be mixed with carbides and nitrides. Also a large amount of the reduced aluminum may be lost as a vapor except in the presence of some alloying agent such as copper or other metals. While it is possible to refine such alloys or mixtures so as to produce commercially pure aluminum, the methods which have been found are too expensive for the present market. One direct reduction method which found limited commercial use in Germany during World War II was the direct reduction of a mixture of clay containing very little

---

R. S. SHERWIN is Vice President, Reynolds Metals Co., Sheffield, Alabama.

AIME New York Meeting, Feb. 1950.

TP 2816 D. Discussion (2 copies) may be sent to Transactions AIME before Apr. 1, 1950, and will be published Nov. 1950. Manuscript received Oct. 20, 1949; revision received Jan. 9, 1950.



iron, with pure alumina in an electric furnace with coke to produce an aluminum-silicon alloy. While the proportion of clay to alumina varied, they were usually mixed in proportions which would produce an alloy containing approximately 60 pct Al and 40 pct Si. In general, these proportions worked better than other mixtures which would produce an alloy higher in aluminum. The product from these furnaces was used as a "mother alloy" or "pre-alloy" to be mixed with aluminum for some of the commercial aluminum-silicon alloys. The maximum production by this method was probably something like 10,000 tons per year.

The commercial production of high grade aluminum from its ores is divided into two separate and distinct processes. The first process produces pure alumina from the ores, and the second process reduces the pure aluminum oxide to metallic aluminum by electrolysis of a solution of alumina in molten cryolite using carbon electrodes.

### Production of Alumina

While many processes have been devised for the production of pure alumina from bauxite, clay, and other minerals, the one which is used predominantly and almost to the exclusion of the others is the Bayer process invented by Carl Joseph Bayer some sixty years ago. Briefly, this process consists of dissolving alumina from bauxite in caustic soda solution to form a sodium aluminate solution, settling or filtering to remove the insoluble residue, precipitating the alumina as hydrate from the solution, and calcining the hydrated alumina to aluminum oxide for use in the electrolytic reduction process.

Among the processes which have been tried was the production of molten alumina in an electric furnace with carbon to reduce the iron, silicon, and titanium to form a heavy alloy in the bottom of the furnace and a molten alumina slag which could be separated, cooled, and used for the production of commercial aluminum by electrolytic reduction. A limited market was found for the iron-silicon-titanium alloy which also contained some aluminum, and the alumina slag was used in the commercial reduction of aluminum. There was some difficulty in obtaining aluminum of the purity desired in the market, and the cost of the process was high so that it was abandoned.

Other processes which have been tried for the production of pure alumina may be classed as acid processes, acid salt processes, and alkaline processes. In the acid processes, the alumina was dissolved from bauxite, clay, or other minerals in solutions of sulphuric acid, nitric acid, hydrofluoric acid, or others. Aside from the cost of these processes, there were difficulties in the complete separation of iron or other impurities, and there was also the difficulty of converting the aluminum salt to alumina by calcination or otherwise.

Among the acid salt processes, the potassium alum process of the Kalunite Co. dissolved the alumina in a solution of sulphuric acid and potassium sulphate to form potash alum which was crystallized, redissolved in a minimum amount of water, heated to precipitate a basic potash alum which was then washed and calcined to form a mixture of potassium sulphate and anhydrous alumina. After removing the potassium sulphate by solution, the alumina was dried for use in the reduction process. Another acid salt process was the ammonium alum method in which clay was treated with ammonium bisulphate to produce an ammonium alum solution. This solu-

tion was then treated with ammonia to precipitate the alumina and convert the ammonium bisulphate to ammonium sulphate. The alumina was calcined for use in the aluminum industry, and the ammonium sulphate solution was evaporated and reconverted to ammonium bisulphate by heating in an electric furnace. This made it possible to recycle both the ammonium sulphate and the ammonia in the process. Both of these acid salt processes were tried during World War II but are not currently in use.

**Alkaline Processes:** A group of alkaline processes have been tried in which high silica bauxite, clay or silicate rocks containing a considerable amount of alumina were heated to incipient fusion with lime or soda or both. There have been so many of these variations that it seems unnecessary to discuss them all. Where lime and soda are both used in the furnace, the lime is proportioned to the silica to form  $2\text{CaO} \cdot \text{SiO}_2$ , and the soda is proportioned to the alumina to form  $\text{Na}_2\text{O} \cdot \text{Al}_2\text{O}_3$ . Where no soda is used in the furnace, the lime is proportioned to form  $2\text{CaO} \cdot \text{SiO}_2$  and 1 to 1.6 mol CaO per mol  $\text{Al}_2\text{O}_3$ . In practice a slight excess of either lime or soda or both is used, and the mixture is heated to temperatures varying from  $1050^\circ$  to  $1200^\circ\text{C}$  or higher, according to the raw materials used or other local conditions.

In the leaching process, enough soda is added to give anywhere from 1.1 to 1.6 mol  $\text{Na}_2\text{O}$  per mol  $\text{Al}_2\text{O}_3$ . This varies according to the degree of stability required in the solution; the higher the proportion of soda to alumina, the more stable the solution will be.

The Pedersen process used a relatively high iron monohydrate European bauxite with limestone and coke in an electric furnace to produce a calcium aluminate calcium silicate slag. This slag was leached with a sodium carbonate solution, usually not over 5 pct, to take the alumina into solution as sodium aluminate. After removal of the insoluble residue by sedimentation, the alumina was precipitated by carbon dioxide, leaving the sodium carbonate in solution for recycling in the process. A plant was built at Hoyanger, Norway, about 1928 to produce about 25 metric tons of alumina per day, and operated up to and during World War II. The iron present in the bauxite was reduced by carbon and sold as pig iron.

One alkaline process developed by the American Nepheline Corporation was tried during World War II near Harleyville, South Carolina. In this process, clay was mixed with limestone and sintered in rotary kilns to produce calcium aluminate, and the sinter was leached with sodium carbonate solutions to produce a sodium aluminate solution. The sodium aluminate solution was then treated with  $\text{CO}_2$  to precipitate the alumina as hydrate and reconvert the soda to sodium carbonate. The aluminum hydrate precipitate was calcined for use in the alumina industry, and the sodium carbonate solution was recycled in the process.

Another alkaline process by the Monolith Midwest Portland Cement Co. sintered anorthosite containing about 27 pct alumina and 3.6 pct  $\text{Na}_2\text{O}$  with lime to produce calcium silicate and sodium aluminate. The sinter was to be leached with sodium carbonate solution to form sodium aluminate solution. The alumina was precipitated by  $\text{CO}_2$  and calcined to produce anhydrous alumina for the aluminum industry. The resulting sodium carbonate



solution was recycled in the process, and it was also proposed to use the calcium silicate residue containing some alumina in the cement industry. A plant for this purpose was built at Laramie, Wyoming, during World War II but was not operated.

The only alkaline furnace process for the production of alumina which is still in operation in this country is a modification of an old process for the treatment of bauxite and other materials containing alumina which had been tried many times. In this process, bauxite containing approximately 50 pct alumina and up to 15 pct silica is first treated by the American modification of the Bayer process, and the insoluble material, usually called red mud, is partially dewatered, mixed with limestone and soda ash, and sintered in rotary kilns. The sintered material is leached with water containing a little soda to produce a sodium aluminate solution. This sodium aluminate solution containing a higher ratio of alumina to soda than Bayer process solutions also contains enough silica to render the alumina precipitate from it undesirable for use in the aluminum industry. For this reason, the solution from the sintering process is recycled to the Bayer process part of the plant so as to precipitate the silica during Bayer process digestion of fresh bauxite. In this way, the solutions from the sinter process are mixed with the Bayer process solution, and the alumina from the combined process is washed and calcined for use in the aluminum industry. The reasons for treating the high silica bauxite by the Bayer process before sintering are that this greatly decreases the load on the sinter plant and avoids the use of a separate process for desilication of the solution as well as avoiding separate precipitation of the sinter solution by means of  $\text{CO}_2$ . Sinter plants for the process were built at four Bayer process alumina plants during World War II, and the one at Hurricane Creek, Arkansas, is still in operation. This combined Bayer and sinter process served to conserve domestic bauxite deposits, and partially relieved the soda ash shortage during and after the war by recovering and returning to the process a large part of the soda which had been lost in the Bayer process due to silica in the bauxite.

The combined Bayer and sinter process seems to be commercially profitable on high silica bauxite where the high silica bauxite is cheap enough to be competitive with low silica imported bauxite and where fuel is cheap and where the sinter plant already exists. It might not be profitable to build a new sinter plant for the purpose, especially when imported bauxite is plentiful.

### Commercial Production of Alumina

While many processes have been devised for the production of pure aluminum oxide, the Bayer process is used predominantly and almost to the exclusion of the others. This process was originally developed in Europe for application to European ores in which the alumina is mainly present as the monohydrate, boehmite. The ground bauxite, either with or without partial calcination, is treated with caustic soda solution containing approximately 350 g NaOH per liter at a temperature approximating  $170^\circ\text{C}$  ( $338^\circ\text{F}$ ) for 2 to 4 hr with a total heating time of 4 to 8 hr so as to produce a solution of sodium aluminate with approximately 1.6 mol  $\text{Na}_2\text{O}$  per mol  $\text{Al}_2\text{O}_3$ . The solution is discharged from the autoclave or digester, diluted to 150-170 g NaOH per liter, and the insoluble residue called red mud is separated by filtration or sedimentation or both. The

mud is washed to remove as much of the soda as is practicable, and the clear solution is cooled to about  $60^\circ\text{C}$  ( $140^\circ\text{F}$ ) and stirred with previously precipitated aluminum hydrate (mainly gibbsite) for three to five days to precipitate 50-60 pct of the alumina from the solution. The resulting precipitate of aluminum trihydrate (mainly gibbsite) is washed to remove soda and then calcined to produce anhydrous alumina,  $\text{Al}_2\text{O}_3$ , for the reduction process. The spent solution is concentrated by evaporation, caustic soda is added to replace losses, and it is then recycled to the autoclaves.

When the production of alumina for the aluminum industry was started in the United States about 1903, it was found that the alumina in American bauxite, mainly gibbsite, trihydrate, was more easily soluble than the monohydrate in European bauxite. Therefore it was possible to work with less concentrated caustic soda solutions and at somewhat lower temperatures. Since the solubility of alumina in caustic soda increases with the caustic concentration and also with increasing temperatures, it is possible to use a variety of combinations of soda concentrations and digestion temperatures. In what we now call the American Bayer process, soda ash is used as a source of the caustic soda, and it is found desirable to combine the lime and soda causticizing reaction with the alumina dissolving reaction in the same vessel. The American Bayer process was standardized so as to use a total soda concentration equivalent to 200-250 g  $\text{Na}_2\text{CO}_3$  per liter, and enough lime is added to the mixture of soda solution and bauxite to raise the total NaOH in the liquor to approximately 113 g NaOH per liter equivalent to 150 g  $\text{Na}_2\text{CO}_3$  per liter. The digestion temperature may vary from  $280^\circ$  to  $290^\circ\text{F}$ , and the time of digestion varies from one-half hour to one hour. Since the total soda in the solution is equivalent to 200-250 g  $\text{Na}_2\text{CO}_3$  per liter, this means that only 60-75 pct of the total alkali is in the form of caustic, and the remainder is carried in the solution as carbonate. Most plants in the United States today probably work with roughly 65 pct of the total alkali causticized to form NaOH. In the usual plant in the United States, the amount of soda converted to NaOH, including that combined with alumina, is called "caustic" or "free soda" and is expressed as grams per liter  $\text{Na}_2\text{CO}_3$  equivalent. When an American alumina plant man speaks of a solution containing 240 g total soda per liter and 150 g free soda or caustic per liter, he means that of the 240 g per liter total alkali expressed as  $\text{Na}_2\text{CO}_3$ , 150 g per liter of this  $\text{Na}_2\text{CO}_3$  equivalent is in the form of caustic soda and sodium aluminate, and the NaOH plus sodium aluminate present is equivalent to about 113 g per liter NaOH, or about  $62\frac{1}{2}$  pct of the total alkali present has been causticized.

In the European Bayer process, the practical limit of caustic soda concentration is just below that concentration which, when nearly saturated with alumina, would be sufficient to salt out some of the alumina and some of the soda as a hydrated sodium aluminate. The caustic soda concentration and the temperature vary somewhat in different alumina plants, but in general the limit of concentration is a little short of that concentration which might salt out sodium aluminates at the lowest temperatures to which strong solutions may be subjected in certain parts of the plant.

In the American Bayer process, the practical limit of caustic soda concentration is that which gives



fairly efficient use of the lime in causticizing without precipitating appreciable quantities of alumina as calcium aluminate. The limiting temperature of digestion is purely economic and varies somewhat in different plants. In general, the temperature and time of digestion should be just sufficient to get the proper amount of alumina into the solution and to precipitate any silica which may have gone into solution as a hydrated sodium aluminum silicate.

In either modification of the Bayer process, the solution after digestion contains 1 mol  $\text{Al}_2\text{O}_3$  to between 1.6 and 1.8 mol free  $\text{Na}_2\text{O}$ . In United States plants, the weight ratio, g  $\text{Al}_2\text{O}_3$  divided by g  $\text{Na}_2\text{CO}_3$ , is usually given instead of the mol ratio. Thus the ratio 1 mol  $\text{Al}_2\text{O}_3$  to 1.6 mol free  $\text{Na}_2\text{O}$  corresponds to our ratio of 0.6 and 1 mol  $\text{Al}_2\text{O}_3$  to 1.8 mol free  $\text{Na}_2\text{O}$  corresponds to our ratio of 0.54 g  $\text{Al}_2\text{O}_3$  to 1 g free soda expressed as  $\text{Na}_2\text{CO}_3$  equivalent.

In the American Bayer process, this does not represent the maximum solubility of  $\text{Al}_2\text{O}_3$  from an uncalcined gibbsite or trihydrate in the bauxite at digestion temperature but represents rather the maximum amount of alumina which can be carried in the solution without serious loss of alumina during separation of the mud by settling or filtration at or slightly below the boiling point of the solution at atmospheric pressure. Approximately the same thing may be said of European practice except perhaps in the case of European bauxite which has been partly dehydrated by calcination before digestion.

These ratios must be regulated carefully for maximum plant capacity without excessive loss of alumina. This is especially true since only about half of the alumina is precipitated before the solution is recycled.

In the Bayer process using either American practice or European practice, the most harmful impurity in the bauxite is silica, especially that which is combined in the form of clay minerals. This combined silica causes serious losses of both soda and alumina. Each pound of combined silica causes a loss of approximately one pound of alumina which might otherwise be extracted and also causes a loss of caustic soda approximately equivalent to one pound of sodium carbonate. For this reason, the value of bauxite for the Bayer process is calculated on the basis of both alumina and silica. Bauxite high in aluminum hydrate and low in silica is regarded as a high grade bauxite while bauxite higher in silica and lower in alumina is considered as a lower grade bauxite. Allowable percentages of silica in bauxite vary greatly, and severe price penalties are put on silica higher than 5 pct.

The loss of soda and alumina is due to the formation of a somewhat indefinite compound or group of compounds which are hydrous sodium aluminum silicates. There is some difference of opinion as to whether there is a definite compound or whether a part of the loss of soda and alumina is due to adsorption on some of these hydrous sodium aluminum silicates. In any case, the chief loss of soda is due to silica in the bauxite, and as the silica in the bauxite increases, the amount of required make-up soda in the process increases rapidly.

We might say that the fundamental difference between European Bayer practice and American Bayer practice lies in the fact that much higher caustic soda concentrations are used in Europe. Since these concentrations are above those which can be produced directly by causticizing sodium

carbonate with lime, it is necessary in Europe either to use caustic soda to make up soda losses or to produce the caustic soda in a separate department of the plant and concentrate it by evaporation to the extent found desirable. The advantage of the original or European Bayer process is that it can treat bauxite in which the alumina is either wholly or mainly in the less soluble monohydrate form called boehmite as well as trihydrate bauxites in which the alumina is more easily soluble. The other form of monohydrate known as diaspor is not suitable for Bayer process treatment due to its low solubility in caustic soda solutions.

Aside from the fundamental difference in caustic soda concentration between European and American practice, there are a number of differences in equipment and operation. Some of these differences are a result of the more fundamental difference in caustic soda concentration. Other differences are due to the difference in labor cost and to more complete mechanization due to the high cost of labor in the United States. Perhaps some of the differences are also due to tradition and to the fact that the plant practice followed different lines of development under conditions where there was little communication in regard to ideas developed in different countries.

We think the American Bayer process is more economical for trihydrate bauxites because it requires less evaporation and because it is more easily adaptable to continuous digestion. American alumina plants originally used batch digestion and indirect heating by means of steam in coils. The digester or autoclave received a batch of solution of suitable volume, and a weighed amount of bauxite was then put in together with the amount of lime required for causticizing and in some cases with an addition of soda ash to replace previous soda losses.

Since the exact concentration of soda is not critical, provided that it is not too high for efficient causticizing with lime, and since the steam coils became coated with scale, the coils were removed and heating was done by direct injection of steam.

The next development was that of adding the required amount of soda ash to the solution before it was pumped into the digester. This batch digestion process as followed at that time involved opening a manhole or head on the top of the digester to receive the charge of bauxite and lime from small cars on a track above the digester. In order to eliminate opening and closing the manhole and to eliminate cars and tracks on the digester floor, the next development was premixing of the batches by dropping the necessary bauxite and lime into a separate batch mixing tank and pumping the mixture to the digestion. Various changes were made in the method of handling, and it was seen that there would be a marked advantage and a saving of steam if the digestion could be made continuous so that the flow of steam from the blow-off would be uniform enough to make the use of heat exchangers more efficient. This resulted in a plan by which a small portion of a metered flow of solution was pumped into a mixing tank, and the bauxite and lime were weighed in small batches on a semi-automatic weighing machine and added to the mixing tank at accurately timed intervals proportioned to the liquor flow so that the proper mixture was maintained at all times. The major portion of the solution was pumped through heat exchangers to the digestors. Thus there is a continuous flow of solution into the



mixing tank and a continuous flow out of the mixing tank to the digester, and the amounts of bauxite and lime are proportioned to the metered flow of solution. By connecting a number of digestors in series with piping and valves and maintaining a uniform flow of a uniform mixture of bauxite and soda solution, it was possible to do the necessary heating by injecting steam continuously through thermostat-controlled valves into the first digester in the series. The required time of digestion was secured by proportioning the flow of the slurry to the volume of the digester series while maintaining a practically uniform pressure and temperature throughout the series of digestors. A common arrangement is six of these digester tanks in a unit with usually five in operation at any one time. The steam piping is connected to the first and second in the series in the unit, but only one of these steamers is used at a time, this being the first one in operation in the series. This made it possible to control temperatures, pressures, and time of digestion and at the same time to blow the solution from the last one in the series at a uniform rate so that the evolution of steam due to reduction in pressure at the blow-off would be practically uniform. By releasing the pressure in two or three stages at decreasing pressures, the steam is used in a series of tubular heat exchangers at two or three different pressures and temperatures. By maintaining a continuous flow of solution through these heat exchangers counter-current to the steam flow, considerable economy in the use of steam was effected.

In some of the more recent American alumina plants, the bauxite is ground wet in ball mills in closed circuit with Dorr classifiers. The bauxite is fed to the mills at a rate suitable for the metered flow of solution used. Only a part of the solution enters the grinding circuit, and the major portion is pumped through the heat exchangers where it is heated first with exhaust steam from the digester blow-off and then with higher pressure steam so as to decrease the amount of steam to be injected in the digestors.

The residue, called "red mud" regardless of its color, is allowed to settle in Dorr thickeners, and the overflow liquor is clarified by filtration through cotton cloth in Kelly filters.

The mud in the underflow from the primary thickeners is washed usually in five or six compartment washing thickeners using five stages of counter-current decantation.

**Evaporation:** Evaporation in the Bayer process is required to remove water equivalent to the amount of wash water used on the waste mud and on the aluminum hydrate except for a decrease due to addition of dry soda and some evaporation incidental to blow-off, etc. In the European Bayer process where the strong solutions used in the digester are diluted with water before filtration or sedimentation, this increases the amount of evaporation required although the dilution water may be kept to a minimum by carrying out the precipitation step at the maximum concentration. Also in the case of European practice where the soda losses are made up by causticizing soda ash with lime in a separate department of the plant, any water introduced in the causticizing operation must also be evaporated.

The evaporation is usually done in multiple effect long tube vertical evaporators. Either all or a part of the solution returning to the digestors may be fed to the evaporators. Since the solubility of sodium carbonate decreases rapidly with increasing caustic

concentration, the limiting concentration of the evaporator discharge in American practice is kept below that at which clogging or serious scaling of the tubes with sodium carbonate would occur. If the tubes do not become clogged, the sodium carbonate scale can be dissolved by changing the flow so that weaker solutions pass through the scaled tubes. In European plants using strong caustic, the solubility of sodium carbonate is so low that the percent of total soda causticized is kept at about 90 pct. Salting out evaporators are not commonly used in the industry, one reason being that the salts do not tend to form crystals large enough to be separated easily. Under certain conditions, salting out may cause the solutions to gel.

Bauxite which was discovered a few years ago in Haiti appears to be a mixture of monohydrate and trihydrate and appears to be intermediate in solubility between the bauxite found on the American continents and that found in Europe. Bauxite in Jamaica in general seems to contain somewhat less monohydrate and somewhat more trihydrate than in Haiti, and there are some orebodies in Jamaica which seem to be almost entirely trihydrate. If a new plant were designed to operate on the general run of bauxite from these two islands, it might be advisable to use caustic soda concentrations intermediate between typical American practice and typical European practice. The choice of the soda concentrations would necessarily be tied in to some extent with the choice of digestion temperature. Some of the orebodies in Jamaica are soluble enough so that they can be used efficiently and satisfactorily by the American Bayer process, and it is likely that these orebodies may be the first ones used in this country.

**Precipitation:** Alumina precipitation in the Bayer process is carried out by cooling the solution and stirring it for a long time with previously precipitated aluminum hydrate. The amount of this hydrate used as "seed" may vary from 25-100 pct of the amount of alumina in the solution. The particle size may also vary widely. In typical German plants, the precipitation is carried out at a concentration of approximately 150 g per liter NaOH and approximately 116 g per liter  $\text{Al}_2\text{O}_3$ . It is common practice to cool the liquor before precipitation to about 60°C (140°F) although in some cases in order to obtain finer particles to use for seed hydrate, the solution may be cooled to 40°C (104°F). The seed hydrate is then added, and the mixture is stirred continuously by wing-shaped paddles on a vertical shaft for from 3 to 5 days in cylindrical flat-bottom tanks about 6 m in diam and 16 m high, during which time the temperature decreases due to radiation and evaporation. Submerged airlifts have been tried experimentally in Europe in these tanks, and some operators are of the opinion that they would be very satisfactory if the tanks had cone bottoms.

In the American Bayer process precipitation, the solution contains initially 100-106 g NaOH per liter and approximately 80 g alumina per liter. It is commonly cooled to 150°-160°F, the seed hydrate is added, and the mixture is stirred for something like 45 hr by means of a submerged airlift in tanks approximately 60 ft deep and 20 to 24 ft in diam with a cone bottom. Thus the American precipitator is for all practical purposes what is known in the mining industry as a pachuca tank.

Regardless of what modification of the Bayer process is used, the aluminum hydrate precipitate must be washed thoroughly to reduce the amount



of soda in the alumina. It is then calcined in rotary kilns using either oil or gas as a fuel. Powdered coal such as is used in the cement industry is not used in the calcination because of the contamination of the resulting alumina with the coal ash.

Various methods of particle size control are used in the precipitation. Low temperature of precipitation and high concentrations in the solution or large amounts of seed of very small particle size tend to reduce the particle size of the resulting product. In this country, the tendency is toward producing alumina particles not larger than approximately 100 mesh and with as little —325 mesh material as possible. Since the precipitate as originally formed contains a considerable quantity of material finer than 325 mesh (44 microns), the hydrate in this country is usually classified in water during the washing operation, and the finer fraction is returned to the precipitators as seed hydrate while the coarser fraction is calcined and used in the production of aluminum metal.

Much of the alumina produced in Europe is finer than that produced in this country, and some of it is so fine that when once dispersed in the air, it settles very slowly. The dust losses, however, are not as great as might be imagined from the fineness of the material because the extremely fine particles tend to stick together and do not disperse readily in the air. In some European plants at the present time, there seems to be a tendency to revert to the very fine particle size formerly used. This is said by some operators to have an advantage in the electrolytic reduction but is open to question, and any disadvantages which one particle size may have over another may be largely overcome by slight changes in the operation of the reduction plant.

### Alumina Specifications

There are no definite specifications for analysis or particle size of alumina for the industry as a whole. Each producing company strives to keep the impurities as low as possible consistent with low costs. Off grade alumina is sometimes segregated for use in production of certain alloys which may require the addition of Si, Fe, Mn, Cu, etc. As far as any definite specifications can be given for the industry in either this country or Europe, we might give the following:

	United States Pct	Europe Pct
SiO <sub>2</sub>	<0.03	<0.10
Fe <sub>2</sub> O <sub>3</sub>	<0.03	<0.10
TiO <sub>2</sub>	<0.002	<0.01
Na <sub>2</sub> O	<0.50	<0.50
Loss on ignition	<0.50	<0.50
		Varies widely with different companies
+ 100 mesh	1.0	0.2
+ 200 mesh	60.	12.
+ 325 mesh	95.	37.
—325 mesh	5.	63. or more

### Commercial Production of Metallic Aluminum

The earliest methods of producing aluminum were by reduction of aluminum chloride with potassium amalgam or potassium, but this soon gave way to reduction by sodium. In 1886, Charles Martin Hall in the United States and Paul Heroult in France, working independently of each other, invented the electrolytic process which is the only one in use commercially today. Anhydrous aluminum oxide of high purity is dissolved in molten cryolite, a double fluoride of sodium and aluminum. The solution of aluminum oxide in molten cryolite

is electrolyzed in the cells or pots of iron lined with carbon so that the carbon lining serves as the cathode. The anodes are also of carbon. The cryolite used as a solvent for alumina is either natural cryolite from the only commercial deposit known, that in Greenland, or synthetic cryolite which is usually made from fluorspar and sulphuric acid to produce hydrofluoric acid, and aluminum hydrate as a source of alumina, and soda ash to furnish the sodium.

Cryolite melts at a temperature of nearly 1000°C. As the alumina is dissolved up to approximately 15 pct, the melting point of the mixture falls to something below 940°C. Further increase in the amount of alumina in solution causes a rapid rise in the melting point. In addition to cryolite, a small amount of fluorspar up to 10 pct may be used to reduce the melting point slightly. A small amount of aluminum fluoride is added to neutralize residual Na<sub>2</sub>O present in the alumina and replace losses of fluorine. Sodium fluoride or soda ash may also be added in small amounts when necessary to control the pH and the melting point. The cell for the electrolytic production of aluminum consists of a strong steel box usually rectangular in shape, provided with a carbon lining 6 to 10 in. or more in thickness. The size of the cell and the size of the electrolyte cavity vary, depending on the number of amperes to be used in the cell. The smallest commercial cell probably does not go below 8000 amp and on general principles, the more current that can be used in a cell, the lower the cost of producing a pound of aluminum. Many cells were built for 30,000 amp but the size has been increased to take as much as 50,000 or 60,000 amp in more recent plants, and a few cells have been built for as much as 100,000 amp.

The carbon lining of the cell is made from a mixture of coke, crushed to suitable sizes, with tar and pitch. This mixture is either rammed and baked in place, or a large portion of the lining may be built up of previously baked carbon blocks with the joints between the blocks filled with a mixture of carbon and pitch.

The carbon anodes are made from a mixture of high grade coke, usually petroleum coke or pitch coke very low in ash with pitch and tar. The anodes may be either pre-formed in blocks in a hydraulic press and then baked, or a continuous self-baking anode may be used. In the case of the Soderberg continuous electrode, a similar mixture but with somewhat more tar is put into a mantle or mold directly above the cell. The electrode mix at the lower end is gradually baked by the heat of the cell as well as by the current passing through the partly baked mix, and is fed gradually into the cell where it is consumed at the bottom in the electrolyte during electrolysis.

The older continuous Soderberg anodes received the current through iron pins inserted through the side of the upper part of the anode mix before baking. As the anode moves downward, the pins in the baked zone are connected to the bus bars. As they move farther down, these pins are disconnected and pulled out before coming in contact with the electrolyte, and the bus bars are connected to the next row of pins above.

A later development by the same company uses larger and longer vertical contact pins which are connected to the bus bars above the continuous electrode and extend downward through the paste anode mix into the baked portion below. As the



anode moves downward and burns off at the bottom in the electrolytic bath, the pins move with it. When the lower end of a pin is within a few inches of the bottom, it is pulled out by twisting and raising it. A small amount of anode paste is then put into the hole, and the pin is lowered to a position 6 to 12 in. above its former position. Advantages are claimed for the vertical contact pins, but the question still seems somewhat controversial. There still remains some difference of opinion also as to whether prebaked anodes or continuous self-baking anodes are better when everything is considered.

Each cell may take anywhere from 8000 to 60,000 amp or in special cases 100,000 amp, according to the size of the cell, at a voltage per cell, including bus bar losses of  $4\frac{1}{2}$  to  $6\frac{1}{2}$  v, depending on the design of the circuit and the cell, age of cell, anode-cathode distance, and other factors. Since it is not economical to generate electricity at such low voltages, a number of cells anywhere from 30 to 100 or more are connected in a line or series. The line current may be supplied by direct current generators, rotary converters, or rectifiers. The tendency in the last few years has been to go to rectifiers. The power required per pound of aluminum varies with various factors, including the anode-cathode distance and the care with which the cell is operated. Possibly 10 kw-hr per lb of metal may be the average overall consumption of power, including line losses and accessories, although the power actually used in the cell may be 8 kw-hr per lb or slightly less under the best conditions. The consumption of carbon anodes varies a great deal with

the quality of the anodes and the other conditions but in general may be from 6/10 to 7/10 of a lb per lb of aluminum.

The electrolytic cell works more efficiently when the carbon bottom is covered with a layer of aluminum. The reasons for this are somewhat controversial, but it appears that a bare carbon cathode causes the production of somewhat more sodium by electrolysis than a cathode covered with molten aluminum. In practice, the current efficiency may vary between 75 and 95 pct. Under good operating conditions, it usually runs in the high 80's.

The molten aluminum may be drawn out by tapping through the end of the cell, but in present practice it is usually drawn out into a crucible through an iron pipe dipped into the cell with one open end below the surface of the molten metal. This may be done by direct suction or by means of a siphon in which a partial vacuum is used to start the siphon. In this case, the discharge end of the pipe is below the surface of molten metal or molten bath in the crucible. The metal in the crucible is then skimmed to remove excessive bath material with some oxidized metal. In some cases, the molten metal is fluxed in the crucible with nitrogen or chlorine gas, either with or without small quantities of aluminum fluoride or other suitable flux.

The metal, either fluxed or not, may then be poured into iron molds to produce pigs weighing approximately 50 lb each, or it may be transferred to a holding furnace, or transferred molten to other furnaces from which, after suitable conditioning or alloying, it may be drawn out for casting into rolling ingots.

## Note on Surface Diffusion in

# Sintering of Metallic Particles

by N. Cabrera

IN a recent paper Kuczynski<sup>1</sup> studied the rate at which the crack between a metallic plane and a spherical particle of the same material is filled up gradually when heated at temperatures near the melting point. If  $x$  is the radius of contact between plane and sphere (fig. 1), Kuczynski shows experimentally that

$$x^2 = At \quad [1]$$

where  $t$  is the time. Assuming that the mechanism of transport of matter is the volume diffusion of vacant lattice sites, he is also able to prove Eq 1 theoretically. The constant  $A$  contains as a factor the self-diffusion coefficient of the metal and he shows that the values for this coefficient deduced from his experiments yield the same activation energy as that calculated by other methods, which is a confirmation of this point of view.

Kuczynski has also considered theoretically the law of growth of  $x$  which would be expected if surface diffusion were the important mechanism. He suggests that  $x$  should then obey a law of the form  $x^2 = At$ , and claims to be able to decide the mechanism involved from the observed law of growth. The purpose of this note is to prove this incorrect.

On the contrary, when the mechanism is surface diffusion Eq 1 is also obeyed, but with a different constant  $A$ .

Following Kuczynski, we suppose the bottom of the crack (surface  $AA$ ) to have an average radius of curvature  $\rho$ . As a result of this curvature the concentration of adsorbed atoms in this surface,  $n(\rho)$  per  $\text{cm}^2$ , will be smaller, and the concentration of vacant lattice sites in the neighborhood,  $N(\rho)$  per  $\text{cm}^2$ , will be larger than the corresponding concentrations,  $n(\infty)$  and  $N(\infty)$ , for a flat surface. For all practical cases we can write Thomson's formula in the form

$$\frac{n(\infty) - n(\rho)}{n(\infty)} = \frac{N(\rho) - N(\infty)}{N(\infty)} = \frac{2\sigma a^2}{kT\rho} \quad [2]$$

where  $\sigma$  is the surface energy of the metal and  $a$  the interatomic distance.

1. Let us now consider the mechanism of surface diffusion. We suppose the concentration of ad-

N. CABRERA is Docteur ès Sciences, International Bureau of Weights and Measures, Sevres, France.

Technical Note No. 35 E. Manuscript received Nov. 19, 1949.



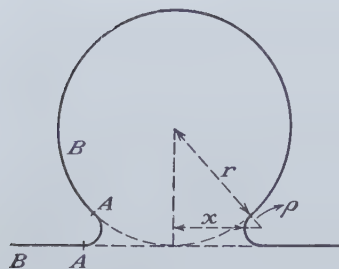


Fig. 1—  
Schematic  
representation  
of the cross-  
section of a  
spherical  
particle  
sintered to a  
metallic block.

sorbed atoms  $n(\rho)$  to be maintained at the bottom of the crack (AA'). As we move towards the flat surfaces BA, the concentration will steadily rise towards the equilibrium value  $n(\infty)$  and there will be a surface current of adsorbed atoms passing any point  $y$  given by

$$j = -D_s \frac{dn}{dy} \text{ cm}^{-1} \text{ sec}^{-1},$$

where  $D_s$  is the coefficient of surface-diffusion for adsorbed atoms, and  $y$  is measured from the points A. On the other hand, as soon as the concentration  $n$  is decreased with respect to the equilibrium value, new adsorbed atoms are formed either by condensation from the vapor, or by evaporation into the surface from kinks<sup>2</sup> existing in the surface. The rate at which the equilibrium tends to be re-established is

$$\frac{n(\infty) - n}{t_s},$$

where  $t_s$  is a relaxation time. In this case the steady distribution of concentration as a function of  $y$  will obey the continuity equation

$$\frac{dj}{dy} = \frac{n(\infty) - n}{t_s} \text{ or } \frac{d^2}{dy^2} [n(\infty) - n] = \frac{n(\infty) - n}{y_s^2} \quad [3]$$

where

$$y_s = (D_s t_s)^{\frac{1}{2}},$$

is a characteristic length for the surface. The solution of Eq 3 is clearly

$$n(\infty) - n = \frac{2\sigma a^3}{kT\rho} n(\infty) \exp\left(-\frac{y}{y_s}\right) \quad [4]$$

where use has been made of Eq 2. The current of adsorbed atoms going through A ( $y=0$ ) per sec per cm is

$$J_s = \frac{4\sigma a^2}{kT} \frac{1}{\rho} \frac{a}{y_s} D_s n(\infty) \quad [5]$$

The product  $n(\infty)D_s$  can be written as

$$n(\infty)D_s = \nu \exp^{-(W_s + U_s)/kT}, \quad [6]$$

where  $\nu$  is the atomic frequency ( $\nu \sim 10^{12} \text{ sec}^{-1}$ ),  $W_s$  the energy of formation of an adsorbed atom from the crystal, and  $U_s$  the activation energy for diffusion. The characteristic length  $y_s$  is difficult to evaluate, but the order of magnitude is  $10^2$ - $10^3 a$  and it increases exponentially as the temperature decreases.

2. We consider now the volume diffusion of vacant lattice points. The steady distribution of concentration will now obey the ordinary Laplace equation and not a Poisson's equation such as Eq 3. The boundary conditions will be  $N = N(\rho)$  near the bottom of the cracks (AA) and  $N = N(y)$  near the practically flat surfaces AB,  $N(y)$  being the con-

centration of vacant lattice sites in equilibrium with the surface concentration  $n(y)$  given by Eq. 4. The problem of course is very difficult, but provided  $x$  is small compared with the radius  $r$  of the sphere (fig. 1), it is probably a reasonable approximation to treat the problem as one with cylindrical symmetry around the cylinder of radius  $\rho$ . The number of vacant lattice points going away from AA, per sec per cm of the axis of the cylinder, is clearly

$$J_v = \pi \frac{D[N(\rho) - N(\infty)]}{\ln \frac{R}{\rho}} = 2\pi \frac{\nu a^2}{kT} \frac{1}{\rho} \frac{a N(\infty) D}{\ln \frac{R}{\rho}} \quad [7]$$

where  $D$  is the diffusion coefficient for vacant lattice sites, and we have used Eq 2. The radius  $R$  is the distance at which  $N=N(\infty)$  and for our purposes it can be taken to be of the order of  $r$ . The quantity  $aN(\infty)D$  can be written as

$$a N(\infty) D = \nu \exp^{-(W+U)/kT} \quad [8]$$

where  $W + U$  is now the activation energy for self-diffusion.

3. It is clear from Eq 7 and 5 that both formulas have the same dependency on  $\rho$ , and therefore they must give the same law for the rate of filling up the cracks. In fact using the equations

$$\rho = \frac{x^2}{4r}, \quad V = \frac{\pi}{4} \frac{x^4}{r}, \quad \frac{dV}{dt} = 2\pi x a^3 J,$$

one obtains easily Kuczynski's Eq 1, in the form

$$\left(\frac{x}{r}\right)^5 = At,$$

$$A_s = \left[ 160 \frac{\sigma a^2}{kT} \left(\frac{a}{r}\right)^8 \right] \frac{a}{y_s} D_s n(\infty), \quad [9]$$

$$A_v = \left[ \frac{80\pi}{\ln \frac{r}{\rho}} \frac{\sigma a^2}{kT} \left(\frac{a}{r}\right)^8 \right] a D N(\infty)$$

The temperature independent factors in  $A_s$  and  $A_v$  are essentially the same. On the other hand, the activation energy for self-diffusion  $W + U$  appearing in Eq 8 is not necessarily bigger than the activation energy appearing in  $(a/y_s)D_s n(\infty)$ , especially if one takes into account a probable exponential factor coming from  $a/y_s$ , which is difficult to evaluate. Kuczynski's experimental results seem to prove that  $A_v \gg A_s$ , in any case for large values of  $r$ .

4. Kuczynski also studied the behavior of small particles (smaller than  $40 \mu$  diam). In this case he obtained, especially at low temperatures, deviations from the Eq 1. In one case the experiments seem to suggest a law of the type  $x^7 = At$ . On the other hand the activation energy which he deduces from the study of the constant  $A$  as a function of temperature coincides with that obtained for bigger particles. It is probably necessary to have more experimental information in order to decide whether a law different from Eq 1 is valid for small particles. From the theoretical point of view we can not expect the rather simple preceding calculations to be valid for small particles, in particular the derivation of the volume diffusion term should require a more careful study.

## References

- 1 G. C. Kuczynski: *Trans. AIME* (1949) **185**, 169. *Jnl. of Metals*, Feb. 1949.
- 2 W. K. Burton and N. Cabrera: *Disc. Faraday Soc.* (1949) **5**, 33.



# Water Sealed Wind Boxes

## for Dwight and Lloyd Sintering Machine

by E. McL. Tittmann and E. A. Hase

Double roasting of sinter carrying a high percentage of lead concentrates, gave rise to the problem of removing the sheets of metallic lead formed in the wind boxes. The solution of the problem has been found in the water sealed wind boxes. The metallic lead is granulated and can be removed without seriously interfering with the operation of the machine and eliminates necessity of men going in wind boxes with air guns.

THE practice of double sintering the charge for lead blast furnaces, which is now standard throughout the lead smelting industry, has introduced the rather difficult problem of cleaning the wind boxes on sintering machines used for the final pass. Two factors have more recently accentuated this problem: (1) the growing demand of workmen for better working conditions and less arduous labor, and (2) the growing dearth of crude lead ores and the improved grade of lead concentrates demand a charge carrying the maximum percentage of lead.

One to 2 pct of such high lead charges (30 to 40) when processed on machines in reasonably good condition will pass through the grates and report in the wind boxes. One-half to three-quarters of this material will be metallic lead reduced by the coke breeze or other fuel mixed with the charge. This lead forms large cakes or chunks which are hard to remove and which substantially increase the time the machine is out of operation for cleaning the wind boxes. A high lead fall can reduce the operating time 30 pct. As a specific example, a 30 ft machine having 63 in. pallets, and equipped with three conventional wind boxes, requires 3 to 4 hr per day for removing the accumulation in the wind boxes when operated as a final pass machine. This same machine when operated on raw charge requires less than an hour per day for cleaning.

To eliminate formation of large chunks of lead in the wind boxes of the final pass machines and time lost in cleaning the boxes, R. C. Rutherford of the

Chihuahua smelter of the American Smelting and Refining Co., devised and patented\* a wind box

---

\* U.S. Patent No. 1,954,951

having a water bath to granulate the lead as it falls from the grate bars. Later, C. H. Harris of the San Luis plant made and patented† an improvement on

---

† U.S. Patent No. 2,348,294

Rutherford's device in which the water bath seals the wind box in a manner permitting the latter to be readily cleaned without destroying the vacuum. The first machine to our knowledge to use the water bath box was at the Chihuahua plant along the lines of the Rutherford patent. In 1942, a machine was placed in operation at the El Paso plant which incorporated the features of both Rutherford and Harris.

This machine is 40 ft long equipped with pallets 63 x 24 in., and having 4 wind boxes 10 ft 4 in. x 7 ft 2 in. x 51 ft 3 in. in size (fig. 1). The air from the fourth box is returned to the first three, and the gas from the first three discharged to a baghouse. The machine has conventional drive, discharge hopper,

---

*E. McL. TITTMANN is Manager, and E. A. HASE is Asst. Sup't., El Paso Smelting Works, El Paso, Texas. AIME New York Meeting, Feb. 1950.*

*TP 2819 D. Discussion (2 copies) may be sent to Transactions AIME before May 1, 1950, and will be published Nov. 1950. Manuscript received Oct. 20, 1949.*



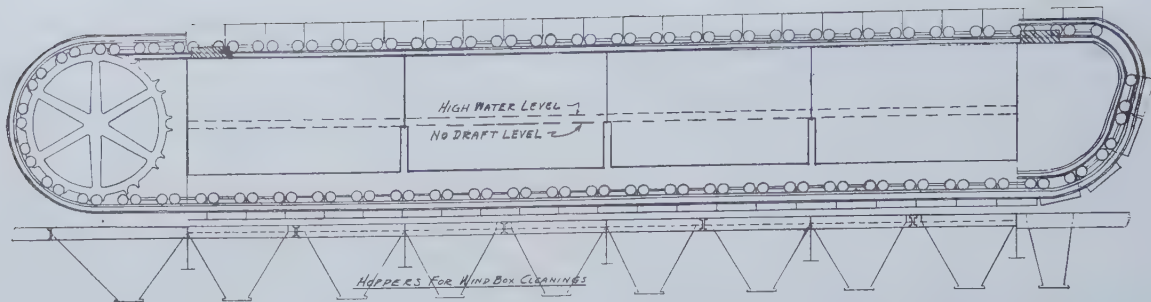


Fig. 1—Longitudinal section through No. 6 D and L machine.

feed hopper, and feeders (fig. 2). The salient features of this design are: (1) vertical ends and sides to below the water lines which prevent metallic lead from the grates building up on these machines; (2) a rounded bottom to allow for easy removal of the cleanings; (3) a clean-out opening sealed by the bath 10 ft 4 in. x 2 ft 8 in. to allow removal of the cleanings without destroying the vacuum in the box. Two baffles or splash plates, one 12 in. above and the other flush with the bath, run the entire length of the box. These plates are spaced equidistant from the sides and effectively prevent water from being sucked into the bustlepipe. With a vacuum of 11 in. of water there is sufficient free board to prevent water from spilling from the discharge openings should the fan be stopped for any reason.

Wind box cleanings are shoveled from the box into a hopper directly below. This hopper in turn discharges to a car which is transported by a lift

truck to either the raw sinter charge or to the blast furnace direct.

The wind boxes each hold 3500 gal of water. Fifty gal are evaporated every 24 hr. To clean the box and splash plates properly the box is drained every 24 hr with the exception of the first box, which is drained every other day. Frequent changing of the water also keeps acidity low preventing damage to the steel. No soluble salts are produced in any quantity. Wind box water after 24 hr analyzes as follows: copper 0, lead 0, and iron 0.6 pct.

The boxes are cleaned once every 24 hr. This requires 45 min compared to 4 hr on a machine of the same size if equipped with conventional wind boxes. The machine is out of service only while changing the water in the first box. Under favorable conditions the machine can be operated at reduced speed while each of the last three boxes is being cleaned. All the material possible is removed without lowering the water. The box is drained to clean the splash plates and the bottom area along the side opposite the clean-out opening. Since the machine is capable of finishing 800 tons of sinter per day the increased capacity because of using water sealed boxes is therefore in the neighborhood of 100 tons per day.

On the present charge, 20 pct lead, the four boxes are producing 16 tons per day. Six tons of this goes direct to the furnace and 10 tons back to the raw sinter charge. On a higher lead charge cleanings from the last three boxes would go direct to the furnace.

Gas leaving the first three boxes contains 3.5 pct sulphur dioxide. Gas from the fourth box contains 2.0 pct. Only a trace of sulphur trioxide is present. After 48 hr water in the first box contains 0.3 pct acid, probably a mixture of sulphuric and sulphurous acid.

In May 1947, boxes Nos. 1 and 4 were renewed. Box No. 1 was made of stainless steel 18/8 Type 316 S M O with Cb, and No. 4 of 7 pct chrome steel. In Feb. 1949, the machine was overhauled and the two alloy steel boxes found in good condition. At that time boxes Nos. 1 and 2 were replaced with stainless steel. All splash plates were renewed with stainless steel.

While the added first cost of water sealed wind boxes would probably not be justified for first over machines or for sintering low lead charges this type of wind box has proved quite successful for second over or final pass machines at El Paso, and has come up to all expectations.

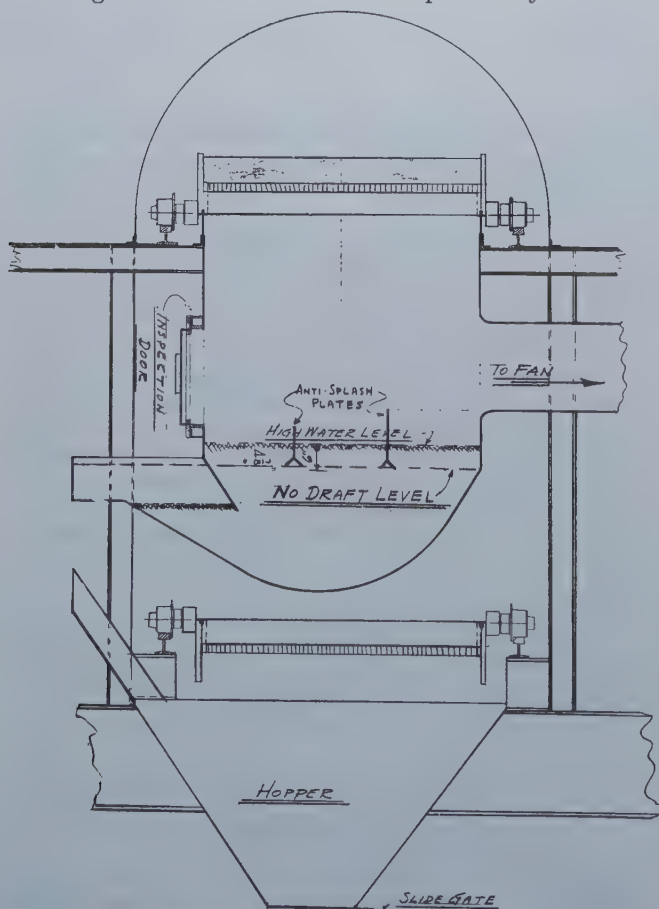


Fig. 2—Cross section through windbox. No. 6 machine.



# Composition of Atmospheres Inert to Heated Carbon Steel

by R. W. Gurry

In a series of charts this paper presents the composition of all gas mixtures, composed of the elements carbon, hydrogen, oxygen and nitrogen, which at temperatures from 1000° to 1800°F are in equilibrium with (1) austenite of various carbon contents, (2) iron and iron carbide, (3) graphite, (4) iron and its oxide.

THIS paper presents the results of calculations made to determine the proportions of CO, CO<sub>2</sub>, CH<sub>4</sub>, H<sub>2</sub>, H<sub>2</sub>O and N<sub>2</sub> in gas mixtures which, at temperatures from 1000° to 1800°F, satisfy, individually, the following conditions: (1) equilibrium with carbon steel of specified carbon content throughout the stable  $\gamma$  iron region; (2) equilibrium with iron and iron carbide, both above and below the eutectoid temperature; (3) equilibrium with graphite; (4) equilibrium with iron and its oxide. From these data a new type of chart has been prepared which permits one to determine, from a room-temperature analysis of the furnace gas in

R. W. GURRY is Physical Chemist, Research Laboratory, United States Steel Corporation of Delaware, Kearny, N. J.

AIME Chicago Meeting, Oct. 1950.

TP 2768 E. Discussion (2 copies) may be sent to Transactions AIME before Dec. 1, 1950, and will be published May 1951. Manuscript received Aug. 16, 1949.

question, the effect of the gas upon the steel at furnace temperature. Thus a guide is provided as to the carbon content which will result and as to the possibility of oxidation or sooting.

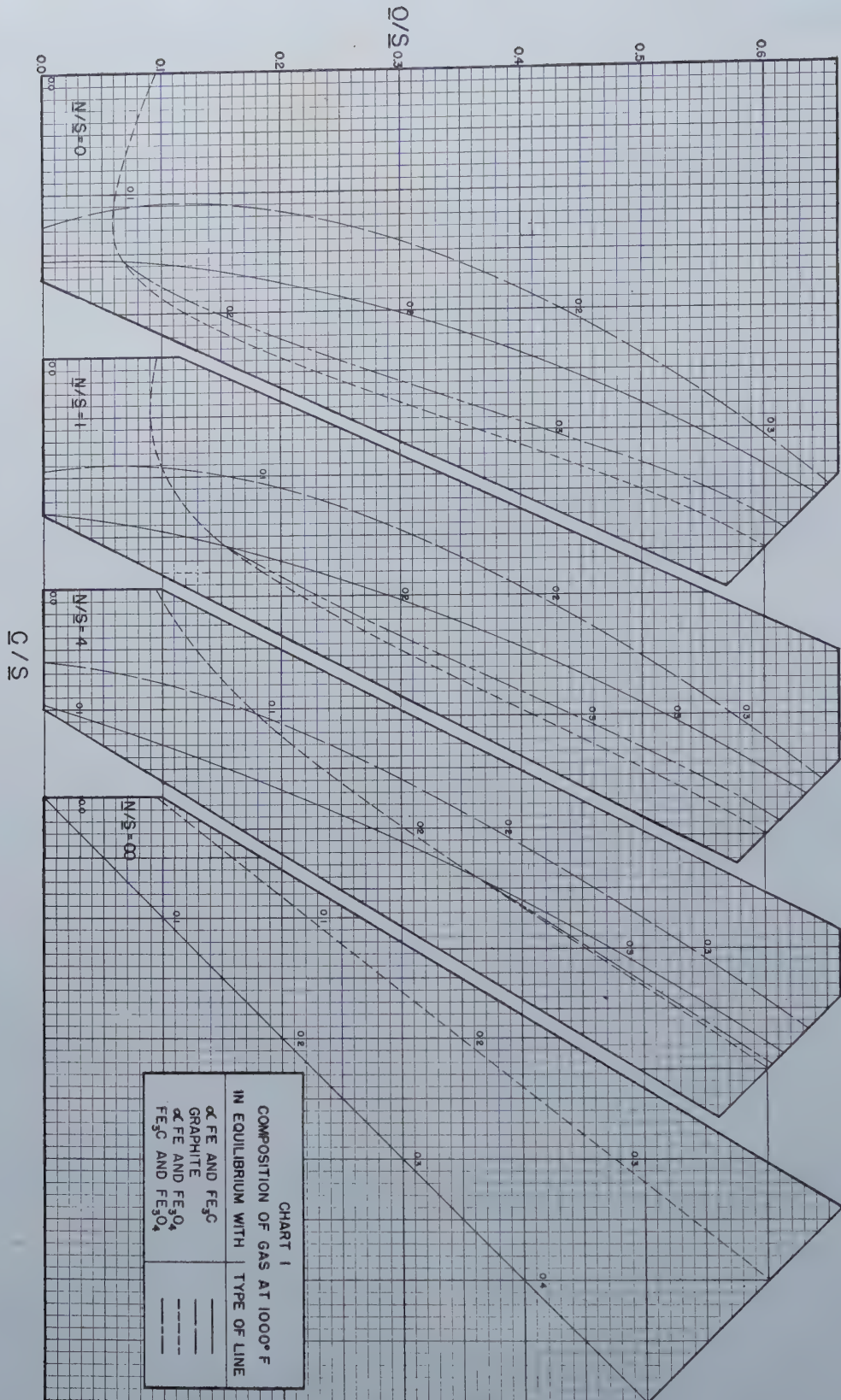
All of the atmospheres commonly used in the commercial heat treatment of steel are composed of some or all of the simple constituents CO, CO<sub>2</sub>, CH<sub>4</sub>, H<sub>2</sub>, H<sub>2</sub>O and N<sub>2</sub>, and also in some cases oxygen or higher hydrocarbons. As the gas mixture is heated certain alterations in the proportions of the constituents take place, and at the temperature of the furnace the composition is, in general, not at all that of the entering gas. If equilibrium in the gas phase is presumed a calculation involving the use of certain fundamental data predicts for a certain temperature the carburizing, decarburizing, or sooting characteristics of a given atmosphere, and, upon the additional assumption of equilibrium between

gas and metal, also the oxidizing ability of the gas and the ultimate carbon content of the steel. The extent to which these predictions are fulfilled will depend upon how closely the assumptions of equilibrium are realized in any particular case. Although at lower temperatures equilibrium is often not reached, there is considerable evidence indicating that above about 1000°F the approach to equilibrium in the gas phase is generally substantially complete, particularly in the vicinity of the steel which acts as a powerful catalyst in promoting reactions in the gas phase. The extent of the gas-metal reactions is commonly affected essentially by the sojourn at temperature; equilibrium with respect to carbon, for example, is accomplished simply by continuing the treatment long enough. However close the approximation of equilibrium may be in any instance, the above mentioned calculation indicates the condition of the steel at equilibrium, and at the present time this is the only basis of control of a furnace atmosphere.

The effect at elevated temperature of a gas of a certain composition at room temperature upon a given steel has customarily been judged by first calculating from gas equilibrium constants the proportions of the gas constituents at the temperature of the steel, then comparing certain of these with the proportions known to be in carbon or oxygen equilibrium. This process is tedious and time-consuming, in particular because the calculation generally involves a series of successive approximations; and since some knowledge and familiarity with this type of computation are required, not everyone needing the information has been, in the past, in a position to obtain it.

By making a series of equilibrium calculations, combining with fundamental data for gas-metal reactions, and plotting the results in a special way, a chart has been prepared which enables anyone

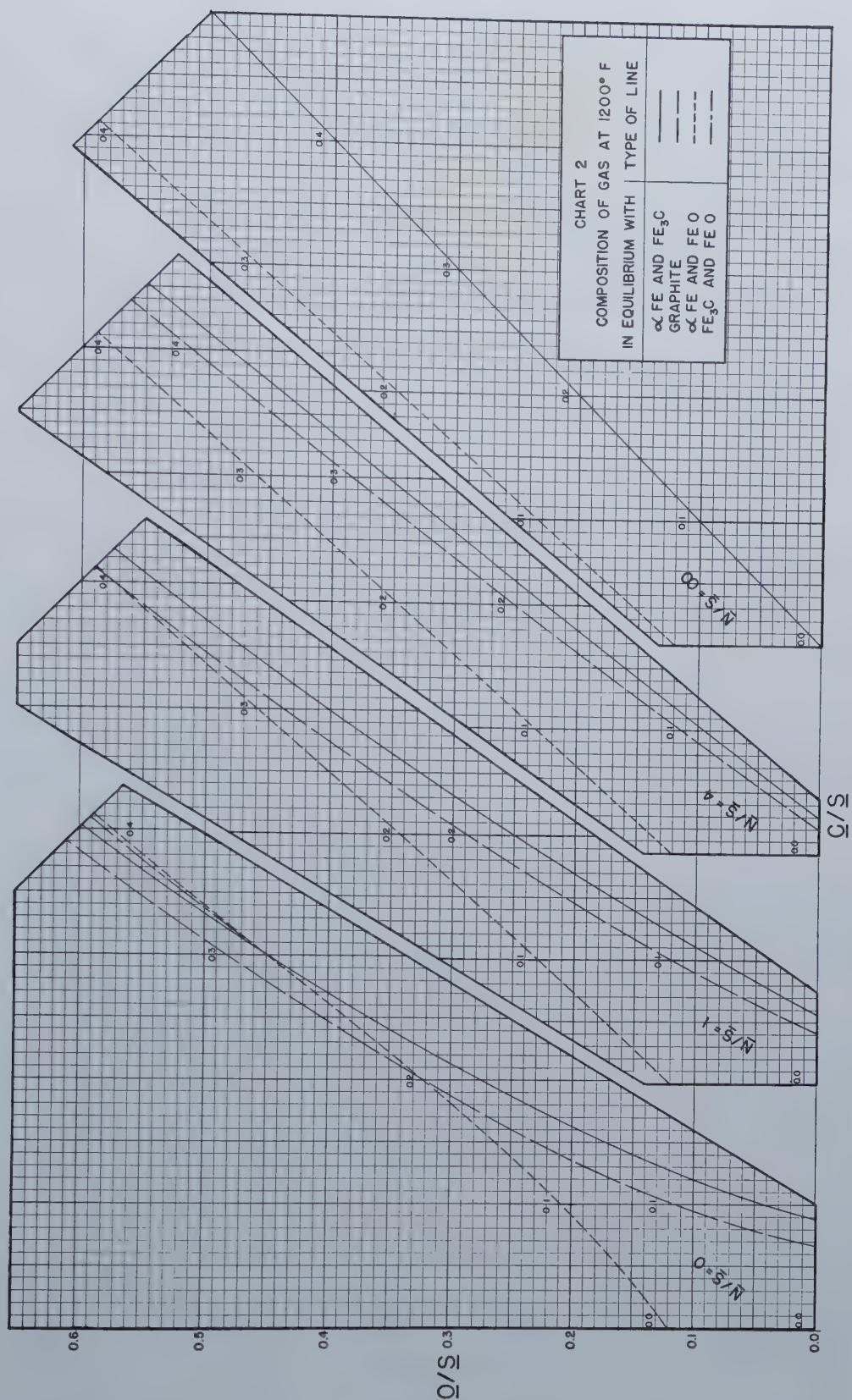




with a rudimentary knowledge of arithmetic to predict from a room temperature gas analysis what the effect of the atmosphere will be upon a steel at furnace temperature under equilibrium conditions. This information not only serves as a guide to the furnace operator but is of particular interest to anyone who must diagnose the cause of unsatisfactory operation of gas producing or heat treating equipment.

Later sections of this paper point out the sources of the data used and the manner in which these data were combined to produce the charts, and also give some general observations from the charts as well as some precautionary suggestions in their application. Since the greater interest will be in the charts themselves, however, they are presented directly, along with sufficient explanation to permit their use.





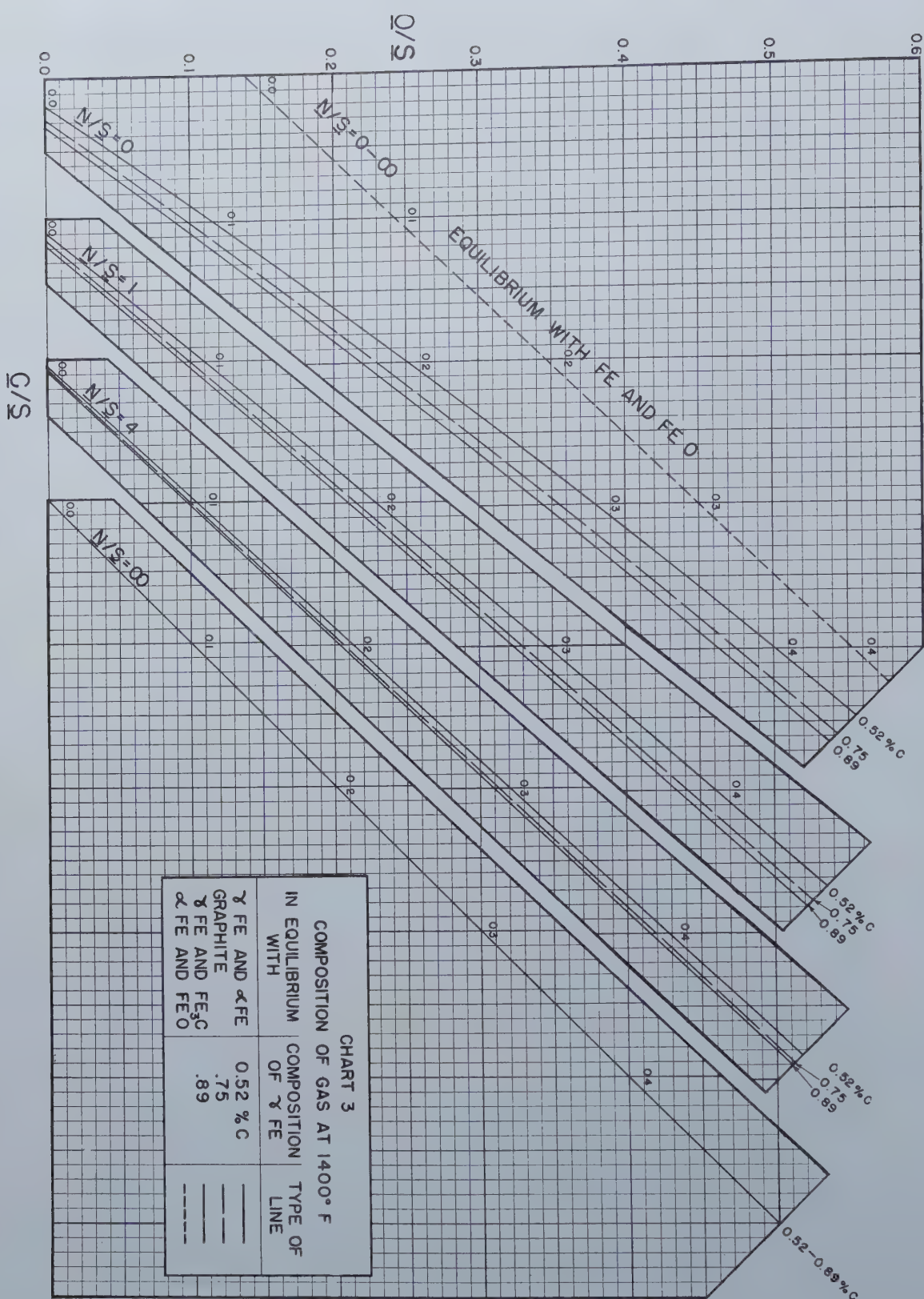
### Use of the Charts

The charts show, on a new type of plot, the gas composition at carbon equilibrium with plain carbon steel and also at equilibrium with oxide. By comparison one can tell directly whether an atmosphere of a given composition is carburizing or decarburizing, oxidizing or reducing, to a given steel at a given temperature, and also whether it is capable of depositing soot. The method of plotting

was designed to give this information quickly and easily; it does not, however, give the molecular constitution of the equilibrated gas at temperature.

As an example, suppose it is desired to find the effect of the gas in a certain furnace upon a given carbon steel. The gas composition to be used should be representative of the atmosphere in actual contact with the steel and is best obtained by inserting a tube into the furnace during opera-

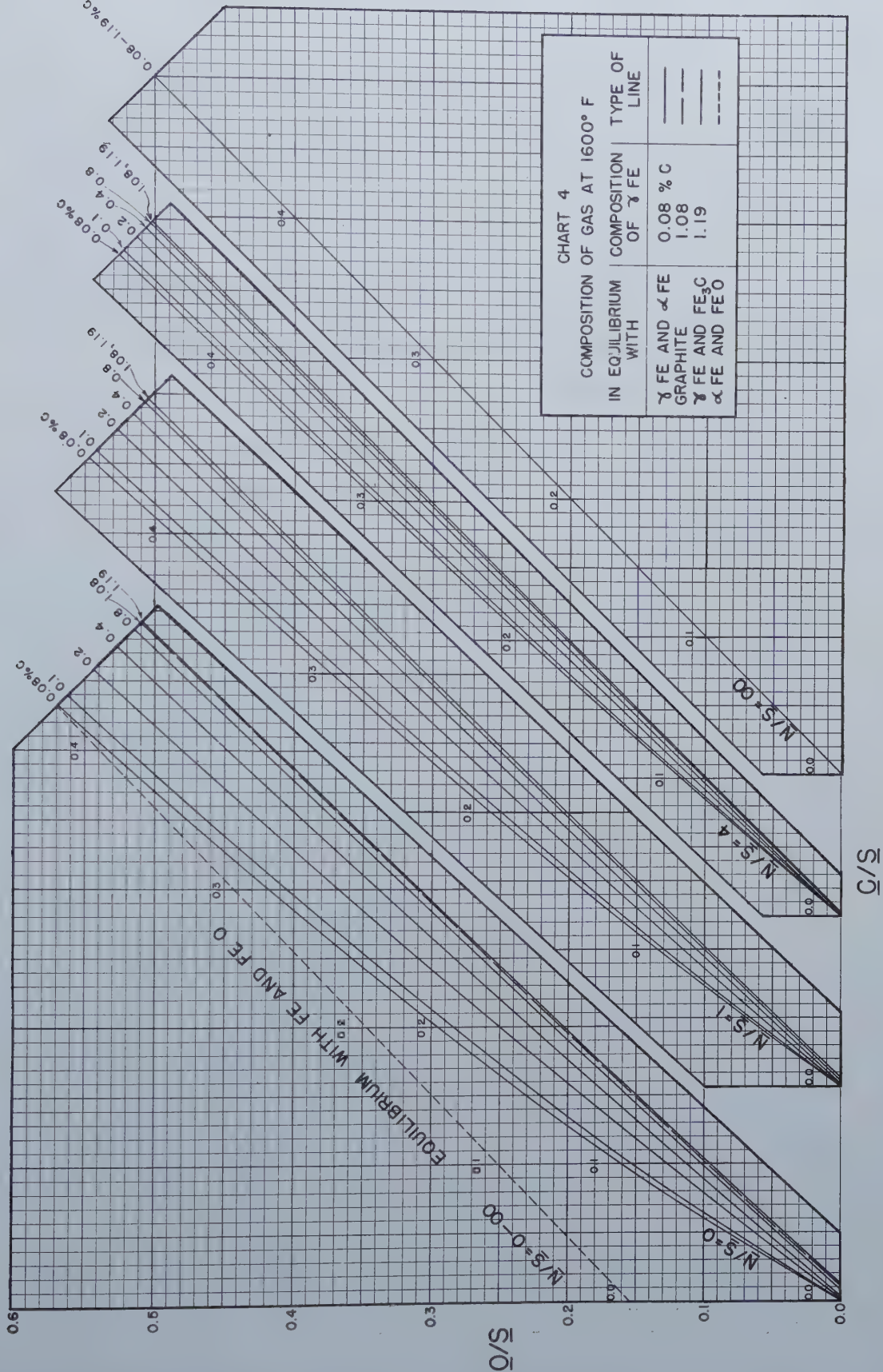




tion and extracting a sample from the vicinity of the work. Another good procedure is to take samples from both the furnace inlet and outlet and carry through the calculation indicated below which will convert each analysis to a point on a chart; if the two points are nearly coincident they may be assumed to represent the gas in contact with the steel providing there are no "dead", or isolated, zones in the furnace. If the two analyses taken from a purged properly designed furnace do not agree

(when converted to the atomic basis, as indicated), then the gas is reacting in the furnace with the steel, with some extraneous material such as oil on the steel, scale, or refractories, or there is a leak in the furnace. After a sufficiently long time, substantial agreement should be obtained between entering and exit gases with a tight furnace. It should also be mentioned that, in order that a gas analysis may give the proper indications, the technique of gas sampling must be correct; long lengths



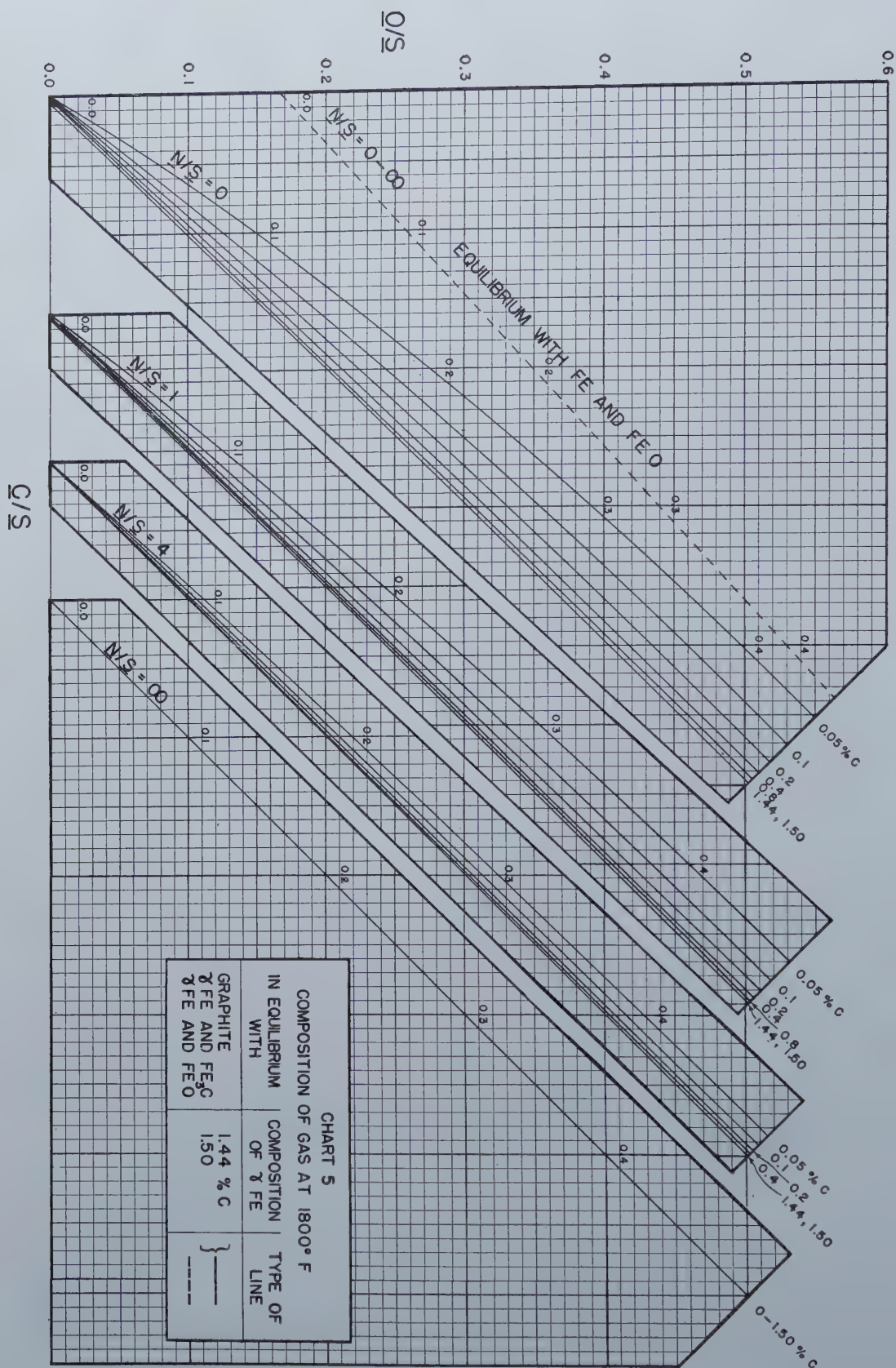


of rubber tubing leading to the sampling bulb, for example, are to be avoided since rubber absorbs a considerable quantity of carbon dioxide. Also the water vapor content must be included in the analysis, and, in the measurement of the dew point, condensation of water in the line must be prevented.

Once a satisfactory gas analysis has been ob-

tained, its effect upon a given steel at a given temperature is indicated by proceeding as follows. Convert the volume percentage of each constituent to the molecular partial pressure by dividing by 100. If water content is given as dew point, this must first be converted to volume percentage by use of a table. The molecular partial pressure in atmos-





pheres is denoted by enclosing the symbol for the constituent in parentheses thus: (CO), (CO<sub>2</sub>), etc. Calculate the "atomic partial pressure" of carbon, oxygen, hydrogen, and nitrogen (denoted  $\bar{C}$ ,  $\bar{O}$ ,  $\bar{H}$ ,  $\bar{N}$ ) using the following relations:—

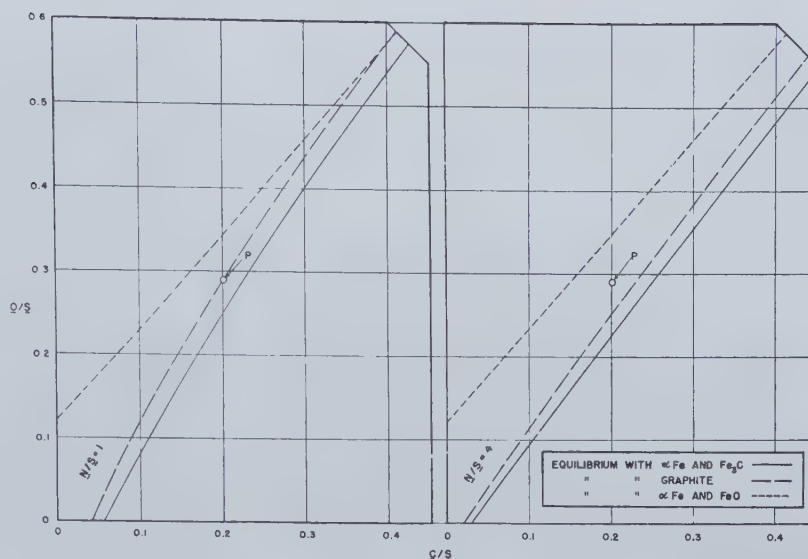
$$\begin{aligned}\bar{C} &= (\text{CO}) + (\text{CO}_2) + (\text{CH}_4) \\ \bar{O} &= (\text{CO}) + 2(\text{CO}_2) + (\text{H}_2\text{O}) \\ \bar{H} &= 2(\text{H}_2) + 2(\text{H}_2\text{O}) + 4(\text{CH}_4)\end{aligned}$$

$$\bar{N} = 2[1 - (\text{CO}) - (\text{CO}_2) - (\text{CH}_4) - (\text{H}_2) - (\text{H}_2\text{O})]$$

If oxygen or hydrocarbons other than methane which will decompose at furnace temperature are present in the ingoing gas corresponding terms should be added. For oxygen the term  $+ 2(\text{O}_2)$  should be added to the expression for  $\bar{O}$  and  $-(\text{O}_2)$  inside the bracket to that for  $\bar{N}$ ; for ethane ( $\text{C}_2\text{H}_6$ ), add  $+ 2(\text{C}_2\text{H}_6)$  to the expression for  $\bar{C}$ ,  $+ 6(\text{C}_2\text{H}_6)$

**Fig. 1—Lines represent gas compositions in equilibrium with the several solid phases at 1200°F.**

Point P is the composition of a furnace atmosphere which, with  $\bar{N}/\bar{S} = 1.67$ , is decarburizing, nonoxidizing and nonsooting.



to that for  $\bar{H}$ , and  $-(C_2H_6)$  inside the bracket to that for  $\bar{N}$ .

Add  $\bar{C} + \bar{O} + \bar{H}$ , designating the sum  $\bar{S}$ , and divide  $\bar{C}$ ,  $\bar{O}$ , and  $\bar{N}$ , in turn by  $\bar{S}$ . The resulting quantities  $\bar{C}/\bar{S}$ ,  $\bar{O}/\bar{S}$  and  $\bar{N}/\bar{S}$ , called the atomic fractions, are the quantities shown on the charts,  $\bar{C}/\bar{S}$  and  $\bar{O}/\bar{S}$  as abscissa and ordinate and  $\bar{N}/\bar{S}$  at chosen increments.

If the point for the ratios  $\bar{C}/\bar{S}$  and  $\bar{O}/\bar{S}$  just obtained plots above and to the left of the curve for the desired carbon content on the chart pertaining to the appropriate temperature and to the  $\bar{N}/\bar{S}$  value just obtained (the  $\bar{C}/\bar{S}$  is less than that which is in equilibrium) the atmosphere is decarburizing to the steel; if it plots to the right and below this curve, the atmosphere is carburizing. Comparison at intermediate temperature, carbon content, or  $\bar{N}/\bar{S}$  value is obtained by interpolation. In a general way, the farther the point is from the line the greater will be the carburizing or decarburizing tendency. If, in addition, it is desired to know whether the given atmosphere tends to oxidize the steel at the temperature, the point corresponding to the atmosphere's composition is compared with the appropriate line denoting equilibrium with oxide. If it falls to the right and below this line (the  $\bar{O}/\bar{S}$  is less than that which is in equilibrium) the steel will not be scaled.

An example will illustrate the necessary calculation and the use of the charts. Suppose it is desired to know the characteristics at 1200°F of a gas of the following analysis:—

	Volume Pct	Partial pressure (atm.)
$H_2$	15.0	0.150
$CO$	10.0	0.100
$CO_2$	5.0	0.050
$H_2O$	3.0 (D.P. 76 F)	0.030
$CH_4$	1.0	0.010

Then:

$$\bar{C} = (CO) + (CO_2) + (CH_4) = 0.100 + 0.050 + 0.010 = 0.160$$

$$\bar{O} = (CO) + 2(CO_2) + (H_2O) = 0.100 + 2(0.050) + 0.030 = 0.230$$

$$\bar{H} = 2(H_2) + 2(H_2O) + 4(CH_4) = 2(0.150) + 2(0.030) + 4(0.010) = 0.400$$

$$\bar{N} = 2[1 - (CO) - (CO_2) - (CH_4) - (H_2) - (H_2O)] = 2[1 - 0.100 - 0.050 - 0.010 - 0.150 - 0.030] = 1.320$$

$$\text{and } \bar{S} = \bar{C} + \bar{O} + \bar{H} = 0.160 + 0.230 + 0.400 = 0.790$$

$$\text{So that } \bar{C}/\bar{S} = 0.160/0.790 = 0.202$$

$$\bar{O}/\bar{S} = 0.230/0.790 = 0.291$$

$$\bar{N}/\bar{S} = 1.320/0.790 = 1.67$$

These three values  $\bar{C}/\bar{S} = 0.202$ ,  $\bar{O}/\bar{S} = 0.291$ , and  $\bar{N}/\bar{S} = 1.67$ , represent the composition of the gas in terms of the variables used in the charts. In fig. 1, which is taken from chart 2, are shown curves depicting the gas composition in equilibrium with the several solid phases at 1200°F. In fig. 1, as in charts 1 to 5, a separate abscissa scale is provided for each  $\bar{N}/\bar{S}$  increment shown; the ordinate scale at the left pertains to all  $\bar{N}/\bar{S}$  ratios. Only the curves for  $\bar{N}/\bar{S}$  values of 1 and 4 are shown, these falling on both sides of the  $\bar{N}/\bar{S}$  of the atmosphere under consideration. The point P on each part of the figure represents the  $\bar{C}/\bar{S}$  and  $\bar{O}/\bar{S}$  ratios of this atmosphere. Since on both parts P lies to the left of the solid curve denoting equilibrium with iron carbide the atmosphere has  $\bar{C}/\bar{S}$  ratio lower than that which is in equilibrium with carbide and is therefore decarburizing. Similarly from reference to the dashed curves it is seen that the atmosphere will not deposit soot since, although P indicates equilibrium with graphite at  $\bar{N}/\bar{S} = 1$ , it is definitely to the left of the graphite equilibrium curve at  $\bar{N}/\bar{S} = 4$ . Ability of a gas to deposit graphite is taken for simplicity as the criterion of its ability to deposit soot, although, strictly speaking, soot, or any other form of carbon, is deposited only at a somewhat higher (but indefinite) activity of carbon.

Tendency of a gas to oxidize is judged by comparison with the dotted curves representing equilibrium with oxide. Since point P lies below the dotted curve in both parts of fig. 1 the  $\bar{O}/\bar{S}$  ratio of the atmosphere is less than the  $\bar{O}/\bar{S}$  ratio for equilibrium with oxide and the atmosphere is classified as reducing. At another temperature the tendencies of the atmosphere will be different. This is indicated by a shifting of the equilibrium curves, the point representing the gas composition remaining constant.

It will be observed that at 1400° and 1600°F the oxidation equilibrium considered is that between  $\alpha$



**Table IA. Activity of Carbon in  $\gamma$ —Iron at Round Carbon Concentration and Temperature**

$^{\circ}\text{F}$ $^{\circ}\text{C}$	1400 760	1500 816	1600 871	1700 927	1800 982	1900 1038	2000 1093
Wt Pct C							
0.05				0.027	0.022	0.018	0.015
0.1			0.067	0.055	0.045	0.037	0.031
0.2			0.139	0.133	0.093	0.078	0.065
0.3		0.269	0.215	0.175	0.144	0.120	0.101
0.4		0.369	0.295	0.240	0.198	0.165	0.138
0.5		0.477	0.382	0.311	0.257	0.213	0.179
0.6	0.756	0.591	0.473	0.385	0.318	0.264	0.222
0.7	0.911	0.711	0.569	0.464	0.383	0.318	0.267
0.8	1.074	0.839	0.671	0.547	0.451	0.375	0.315
0.9		0.977	0.782	0.637	0.525	0.437	0.367
1.0		1.119	0.895	0.729	0.602	0.500	0.420
1.1			1.018	0.829	0.684	0.569	0.477
1.2				0.936	0.773	0.642	0.539
1.3				1.044	0.862	0.716	0.601
1.4					0.961	0.798	0.670
1.5					1.062	0.883	0.741
1.6						0.973	0.816
1.7							0.898
1.8							0.980

Only five charts corresponding to the temperatures 1000°, 1200°, 1400°, 1600° and 1800°F are presented here; and each of these contains information only at N/S ratios of 0, 1, 4 and  $\infty$ . A set of large folding charts, including also those for the temperatures 800°, 1100°, 1300°, 1500° and 1700°F and including the additional N/S increments of 0.5, 2, 8 and 16, has been prepared and is available on request.\*

\* Address: Research Laboratory, U. S. Steel Corp. of Delaware, Kearny, N. J.

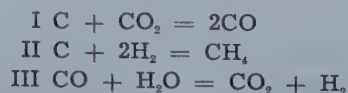
### The Basis of Control of a Furnace Atmosphere

A controlled atmosphere as supplied to a furnace used for the heat treatment of steel generally consists of some or all of the constituents carbon monoxide (CO), carbon dioxide (CO<sub>2</sub>), methane (CH<sub>4</sub>), hydrogen (H<sub>2</sub>), water vapor (H<sub>2</sub>O), nitrogen (N<sub>2</sub>), oxygen (O<sub>2</sub>) and higher hydrocarbons. All the constituents except nitrogen tend to be either

oxidizing or reducing and in addition either carburizing or decarburizing. Nitrogen we consider to be neutral, and regard as affecting the steel only as its pressure alters the partial pressure of the active constituents. When the atmosphere is heated the proportions of the active constituents are altered in accord with a number of chemical reactions which proceed in one direction or the other. For example, the percentage of oxygen or of higher hydrocarbons is reduced to substantially zero. If final equilibrium is presumed the proportion of each constituent is determinate, and can be evaluated by simultaneous solution of several relations

involving the application of previously measured fundamental data.

The interrelation of the constituents may be illustrated by a mechanical analogy. In a nicely balanced system of interconnected levers, each supporting a certain weight, a change of weight on any one lever will, in general, alter the position of each of the others; and the correct calculation of the new equilibrium position of each lever requires the simultaneous consideration of several factors. All possible equations which might be written to interrelate the positions of the levers are, of course, not independent. In a similar way all the reactions involving the active constituents of the gas are not independent; and consideration of a properly selected group of three of them suffices to fix the final state of the whole system. The group of three selected as most suitable for our purpose is:



The first two of these, when read from left to right, correspond to a transfer of carbon from the solid phase, which may be graphite or a steel, to the gas atmosphere; when read from right to left, to a deposition of carbon as graphite or to its solution in a steel resulting in an increase in carbon content of the steel. The third reaction represents the readjustment of composition of the gas mixture occasioned by the occurrence of the primary reactions I and II. Whether, in which direction, and how far, reactions I and II proceed is determined by whether a certain quotient (of quantities relating

**Table IB. Carbon Content and Activity in  $\gamma$ —Iron in Equilibrium with  $\alpha$ —Iron, Graphite or Iron Carbide at Round Temperature**

	$^{\circ}\text{F}$ $^{\circ}\text{C}$	1333 723	1400 760	1500 816	1600 871	1700 927	1800 982	1900 1038	2000 1093
Equilibrium with $\alpha$ -iron	(Pct C ( $a_c$ )	0.800 1.29	0.520 0.640	0.248 0.219	0.082 0.055				
Equilibrium with graphite	(Pct C ( $a_c$ )	0.650* 1	0.754 1	0.917 1	1.084 1	1.260 1	1.439 1	1.628 1	1.821 1
Equilibrium with iron carbide	(Pct C ( $a_c$ )	0.800 1.29	0.894 1.24	1.039 1.18	1.186 1.13	1.342 1.09	1.502 1.06	1.677 1.05	1.865 1.04

\* Extrapolated.

iron and Wüstite ("FeO") although the form of iron of primary interest at these temperatures is  $\gamma$  iron. The reason for this is that in the process of scaling a  $\gamma$  iron-carbon alloy at these temperatures, carbon is first removed and the surface converted to  $\alpha$  iron. Thus the reaction considered is justified; in fact, it is the only one for which equilibrium data can be obtained.

It will be observed that a single set of values of C/S, O/S and N/S, corresponding to a single point on a chart, represents an infinitude of different molecular compositions. One might infer from this that the molecular constitution of the gas supplied to a furnace does matter as long as the atomic composition of the mixture, as indicated by the charts, is right. This inference, of course, is not correct. The rates of the various reactions involved in the equilibration of a gas mixture differ; and in one particular case, that of sooting, a substance is produced which is taken up again only slowly during the further approach to equilibrium. It is generally desirable to supply to the furnace a gas of a composition as near as possible to that at equilibrium at temperature. It is the purpose of the gas converter to supply such a controlled atmosphere.

If equilibrium is not attained the indications afforded by the charts are, of course, not strictly valid; they are to be regarded as tendencies of the atmosphere, however, and as such serve a useful purpose. It is to be noted that any discrepancy between what is observed and what is indicated by the charts is in no way connected with the method of plotting used here and would occur in any comparison of the observation with the fundamental equilibrium data.

to the three constituents in each of these reactions, to be defined in a moment) is equal to, greater or less than, the so-called equilibrium constant characteristic of each reaction at the temperature of the steel. It is assumed here, of course, that the rate of all the reactions is sufficiently great to permit them to come substantially to equilibrium; this is generally true when the temperature is above about 1000°F, since above this temperature all of these reactions go rapidly, being catalyzed at an iron surface.

The equilibrium constant for each of these three reactions we define, in the first place, as follows in column 1,  $a$  being the thermodynamic activity of the constituent denoted by the subscript:

(1)	(2)	(3)	(4)
			Numerical value at 1400°F*, $p$ expressed in atmospheres
$K_1 = \frac{a_{CO}^2}{a_C a_{CO_2}} = \frac{p_{CO}^2}{a_C p_{CO_2}} = \frac{(CO)^2}{a_C (CO_2)}$			3.66
$K_2 = \frac{a_{CH_4}}{a_C a_{H_2}^2} = \frac{p_{CH_4}}{a_C p_{H_2}^2} = \frac{(CH_4)}{a_C (H_2)^2}$			0.0698
$K_3 = \frac{a_{CO_2} a_{H_2}}{a_{CO} a_{H_2O}} = \frac{p_{CO_2} p_{H_2}}{p_{CO} p_{H_2O}} = \frac{(CO_2)(H_2)}{(CO)(H_2O)}$			1.20

\* From table II.

The activity  $a$  of any of these gases, at the low pressures (less than 1 atm) prevalent in the usual furnace atmosphere, may be replaced by its partial pressure expressed in terms of atmospheres; when this is done, we have the definitions in column 2 above. For convenience of notation, we shall throughout use the form in column 3—that is, the symbol ( $X$ ) is used in place of  $p_X$  to denote the partial pressure of the gas  $X$  in the mixture, the percentage by volume being therefore 100( $X$ ). This leaves as the only activity that of carbon,  $a_c$ ; it may be regarded as the effective tendency of carbon as a constituent of the solid phase to take part in the reaction, as distinct from the gross carbon content measured by ordinary chemical analysis. It is proportional to the vapor pressure of carbon in equilibrium with the solid phase which, however, is difficult to measure directly. To overcome this difficulty we may describe the activity as a quantity, pertaining to the particular solid phase, which, when inserted in the expression for any of the equilibrium constants yields a result in accord with experiment; and it is in this way that a numerical value is assigned to it. This value is not absolute, but always relative to some standard state, this being chosen as the one most convenient for the purpose in hand.

For the present purpose we refer the activity  $a_c$  (or virtual vapor pressure) of carbon in the solid phase to that of graphite as unity, at all temperatures. This simplifies the treatment when a number of different temperatures is to be considered. The activity of carbon in a steel which has come to equilibrium with graphite—that is, is saturated with respect to graphite as second solid phase—is by definition likewise unity. In the more usual case of a steel not saturated with graphite,  $a_c$  is determined in the following way:

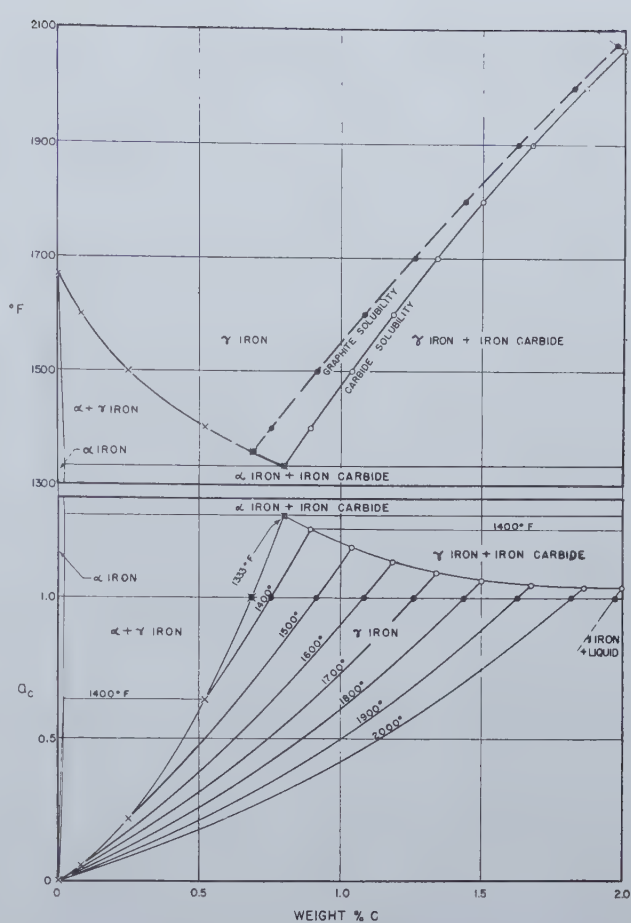


Fig. 2—Plot of temperature and carbon activity ( $a_c$ ) against carbon content. The solid circles, the hollow circles and the crosses in the lower diagram correspond to similar points in the upper diagram, and indicate equilibrium of  $\gamma$  iron with graphite, iron carbide and  $\alpha$  iron, respectively.

**Determination of  $a_c$  in a Steel:** When carbon monoxide, or carbon dioxide, or any mixture of them, alone or with an inert gas, is passed slowly over pure graphite maintained at a constant temperature—which for the sake of definite figures, we shall take at 1400°F—and the outgoing gas is analyzed so as to determine the partial pressure of each of the two active constituents, we find that the quotient of partial pressures  $(CO)^2/(CO_2)$ , has the numerical value 3.66 no matter what the ingoing mixture was; consequently, since  $a_c$  is here by definition unity,  $K_1 = 3.66$  at 1400°F. Now if a precisely similar experiment is carried out in which (1) graphite is replaced by a steel, and (2) the ingoing gas is replaced by a constant mixture of CO and  $CO_2$  in initial proportions such that the quotient  $(CO)^2/(CO_2)$  in the gas is equal to 3.66, and if this experiment is continued until the composition of the outgoing gas (and the carbon content of the steel) remains constant the carbon content of the steel will then be 0.75 pct, the composition in equilibrium with graphite ( $a_c = 1$ ) at 1400°F (table I). If the experiment is repeated using a gas whose  $(CO)^2/(CO_2)$  is other than 3.66, the corresponding value of  $a_c$  is obtained directly by substitution of the partial pressures in the expression for  $K_1$ . As a specific instance, if (CO) and  $(CO_2)$  are initially 0.78 and 0.22 atm respectively, the activity of car-



Table II. Selected Values of the Several Equilibrium Constants

Temperature		Equilibrium with carbon		(CO <sub>2</sub> ) (H <sub>2</sub> )	Equilibrium with iron and its oxide*		Activity of carbon in iron saturated with iron carbide†
		(CO) <sup>2</sup>	(CH <sub>4</sub> )		(CO <sub>2</sub> )	(H <sub>2</sub> O)	
		<i>a<sub>c</sub></i> (CO <sub>2</sub> )	<i>a<sub>c</sub></i> (H <sub>2</sub> ) <sup>2</sup>	(CO) (H <sub>2</sub> O)	(CO)	(H <sub>2</sub> )	
°F	°C	K <sub>1</sub>	K <sub>2</sub>	K <sub>3</sub>	K <sub>4</sub>	K <sub>5</sub>	<i>a<sub>c</sub></i> '
800	427	0.000267	8.97	9.02	1.21‡	0.134‡	6.18#
900	482	0.00232	3.07	5.63	1.10‡	0.196‡	4.08#
1000	538	0.0155	1.18	3.75	1.01‡	0.270‡	2.88#
1100	593	0.0783	0.516	2.66	0.885	0.333	2.14#
1200	649	0.333	0.245	1.97	0.750	0.381	1.66#
1300	704	1.18	0.126	1.51	0.651	0.430	1.37#
1400	760	3.66	0.0698	1.20	0.575	0.478	1.24
1500	816	10.1	0.0406	0.980	0.518	0.528	1.18
1600	871	24.9	0.0251	0.820	0.472	0.575	1.13
1700	927	57.2	0.0161	0.697	0.435	0.624	1.09
1800	982	120.	0.0107	0.604	0.405	0.670	1.06
1900	1038	238.	0.00740	0.530	0.380	0.718	1.05
2000	1093	440.	0.00527	0.472	0.357	0.757	1.04
2100	1149	783.	0.00383	0.423	0.338	0.800	
2200	1204	1320.	0.00286	0.383	0.321	0.839	

\* Below 560°C (1040°F) the stable oxide is magnetite (Fe<sub>3</sub>O<sub>4</sub>), data marked with the sign ‡. At higher temperatures it is wüstite (FeO<sub>x</sub>, where *x* has a value slightly greater than unity depending upon temperature).

† The form of iron below 723°C (1333°F) is  $\alpha$ -iron, data designated with the sign #; over the range 723°–1130°C (1333°–2066°F) the form is  $\gamma$ -iron.

bon both in the gas and in the steel after equilibration at 1400°F is obtained thus:

$$a_c = \frac{(\text{CO})^2}{K_1 \times (\text{CO}_2)} = \frac{(0.78)^2}{3.66 \times 0.22} = 0.76$$

and the final carbon content will be 0.60 pct\* by weight. In the case of a gas in which (CO)<sup>2</sup>/(CO<sub>2</sub>) is greater than 3.66 held at 1400°F, one might expect that graphite would precipitate and that the ratio by this means would return to the value 3.66; this is, in fact, exactly what would occur after a sufficiently long time. However, this precipitation of graphite within the gas phase is slow relative to equilibration of the gas with the carbon in the steel and carburization can be carried, by a suitable gas mixture, up to saturation with iron carbide where *a<sub>c</sub>* is 1.24 and the equilibrium carbon content 0.89 (at 1400°F)\*.

\* From table I.

Systematic series of such measurements, in which the composition of the constant ingoing mixture was changed by definite steps, at several constant temperatures, have been made.<sup>1</sup> Table IA derived from these data gives directly for each of several constant temperatures the numerical value of *a<sub>c</sub>* in a carbon steel at each 0.1 pct carbon over the  $\gamma$ -iron field; from it *a<sub>c</sub>* for any intermediate carbon content can readily be interpolated. Similar values of *a<sub>c</sub>* and the corresponding carbon content at the boundaries of the field, and also the carbon content in equilibrium with graphite (*a<sub>c</sub>* = 1) are given in table 1B; these were obtained by combination of the activity data with a revised iron-carbon diagram, presented in fig. 2 and discussed below. It should not be inferred that either the activities or the odd percentages given in table I are accurate in the third decimal place; they do, however, constitute a consistent set as given, and for this reason they are recorded here and used in this paper. The data in this table are strictly valid only for pure iron-carbon alloys, but are substantially accurate for ordinary carbon steels; they would not be even approximately valid for steels carrying moderate proportions of most other elements. The difference

in numerical value of *a<sub>c</sub>* at a given carbon content, e.g. 0.5 pct, at different temperature levels is due to the difference in solubility of graphite in iron; if *a<sub>c</sub>* were relative to the activity in infinitely dilute solution as unity, it would not vary appreciably with temperature.

In fig. 2 we present the selected iron-carbon temperature-composition equilibrium diagram (*upper portion*), and for direct comparison a plot of carbon activity (*a<sub>c</sub>*) against carbon content (*lower portion*). The carbon content of  $\alpha$  iron in equilibrium with  $\gamma$  — iron is from the work of Smith,<sup>1</sup> that of  $\gamma$  — iron in equilibrium with  $\alpha$  — iron from Mehl and Wells.<sup>2</sup> The line representing saturation of  $\gamma$  — iron with respect to graphite is based on the measurements, made by different methods, of Wells,<sup>3</sup> Smith<sup>1</sup> and Gurry.<sup>4</sup> From this line, the corresponding line representing the limiting solubility of iron carbide in  $\gamma$  — iron was derived by means of thermodynamic relations between:

- (1) the free energy change of the reaction  $3\text{Fe}(\gamma) + \text{C}(\text{graphite}) = \text{Fe}_3\text{C}$ , as evaluated by Darken, Smith and Gurry (*to be published*);
- (2) the activity of carbon in  $\gamma$  — iron, as measured by Smith<sup>1</sup>;
- (3) the cementite eutectoid temperature and composition as given by Mehl and Wells.<sup>2</sup>

The activity isotherms throughout the  $\gamma$  — iron field in fig. 2 are plotted directly from the data in table I, which are derived from Smith's data<sup>1</sup> by use of the graphite solubility at each temperature as the standard of reference. The activity at points along each boundary between fields corresponds to the carbon content as read from the upper diagram. The 1400° isotherm is extended into the adjacent fields in order to illustrate: (1) that at any certain temperature activity increases continuously with carbon content throughout any single phase region, whether the iron is  $\alpha$  or  $\gamma$ ; (2) that the carbon activity of two phases in equilibrium is (by definition) the same, regardless of the relative amount of each phase.

Each point on the graphite solubility line corresponds precisely to one at unit activity at the same temperature and carbon content in the lower part of fig. 2; and points along the several field boundaries in the two parts of fig. 2 likewise correspond. Any iron-carbon alloy containing carbon in excess of the graphite solubility at the temperature, being then thermodynamically unstable, tends to precipitate carbon as graphite until the carbon actually in solution is lowered to the equilibrium concentration; but this precipitation occurs only after a very long time at temperature (except at very high temperatures or when certain elements, e.g. silicon, are present), so that this excess carbon usually remains in solution.

Consequently, carbon activity may exceed unity, as shown in the figure, up to an amount corresponding to saturation with respect to iron carbide; for instance, at 1400°F it is 1.24, and the carbon content is 0.89 pct. This steel is in equilibrium with a CO-CO<sub>2</sub> mixture in which (CO)<sup>2</sup>/(CO<sub>2</sub>) = *K<sub>1</sub>a<sub>c</sub>* =  $3.66 \times 1.24 = 4.54$ ; consequently any gas mixture in which (CO)<sup>2</sup>/(CO<sub>2</sub>) is greater than 4.54 will produce carbide as a separate phase on the surface of the specimen and thereby increase its gross carbon content beyond 0.89 pct which is, of course, still the concentration of carbon in solution.



## Numerical Values of the Equilibrium Constants

In describing how carbon activity,  $a_c$ , is evaluated, we outlined the way in which the equilibrium constant of a reaction is determined. In calculations involving more than one such constant, it is essential that the values selected be concordant with one another and with the constants for related schematic reactions. For example:

$$\frac{K_1}{K_2} = \frac{(\text{CO})^2(\text{H}_2)^2}{(\text{CO}_2)(\text{CH}_4)}, \text{ the constant for the re-}$$

action  $\text{CO}_2 + \text{CH}_4 = 2\text{CO} + 2\text{H}_2$  and its numerical value at  $1400^\circ\text{F}$  is  $3.66/0.0698 = 52.4$ .

The collation and reconciliation of the various data bearing on the selection of the best and most consistent numerical values of these constants have been discussed in a number of previous papers,<sup>5</sup> and will not be taken up again here. For, after studying this whole matter again, we finally decided to make use of the consistent set of constants derived (from calorimetric and spectroscopic data) and published by a group<sup>6</sup> at the Bureau of Standards, even though the value which they assign to  $K_3$  differs somewhat from that calculated from direct measurements of the water-gas equilibrium. This small difference is, however, not material to the present purpose. The values of  $K_1$ ,  $K_2$ , and  $K_3$  given in table II for each  $100^\circ\text{F}$  over the temperature range  $800^\circ$ – $2200^\circ\text{F}$  were interpolated from those published by them by plotting against temperature the deviation from a linear relationship between  $\log K$  and the reciprocal of the absolute temperature.

The ratio  $(\text{CO}_2)/(\text{CO})$ , designated  $K_4$  in table II, at which iron is in equilibrium with its oxide was read from a curve drawn to represent the best data in the literature; and the ratio  $(\text{H}_2\text{O})/(\text{H}_2)$ , designated  $K_5$ , was obtained from  $(\text{CO}_2)/(\text{CO})$  by dividing the latter by  $K_3$ . In the higher part of the temperature range covered the data used were those of Darken and Gurry;<sup>7</sup> at intermediate temperatures the data considered were those of Schenck and coworkers,<sup>8</sup> and of Emmett and Schultz;<sup>9</sup> while at the lower temperatures where iron is in equilibrium with magnetite the latter were the only data available. The values of  $(\text{H}_2\text{O})/(\text{H}_2)$  as measured by Emmett and Schultz were converted to the equivalent  $(\text{CO}_2)/(\text{CO})$  using the water gas constant ( $K_3$ ) as given by Kassel;<sup>10</sup> but corrected in conformity with the newer heat of reaction  $\Delta H_r^\circ$  given by Rossini.<sup>11</sup> This value of  $K_3$  rather than that from table II was used here because it made the data of Emmett and Schultz agree better with those of the other investigators who measured the ratio  $(\text{CO}_2)/(\text{CO})$ .

The activity of carbon in iron saturated with iron carbide is given in the last column of table II. The values above the eutectoid temperature are from table I, their derivation from carbide solubility and activity data having been described in a previous section. Below the eutectoid, the solubility of carbon in  $\alpha$ —iron is so low that the activity of iron in the solution can be considered equal to that of pure  $\alpha$ —iron; and the activity of carbon at saturation with carbide is obtained thermodynamically from the standard free energy change of the reaction  $3\text{Fe}(\alpha) + \text{C}(\text{graphite}) = \text{Fe}_3\text{C}$ , as evaluated by Darken, Smith and Gurry (to be published). The activity of carbon in  $\alpha$  and  $\gamma$ —iron must, of course, be the same at the eutectoid since at this point there is equilibrium between iron carbide,  $\alpha$ —iron,

and  $\gamma$ —iron. Both methods used do, in fact, give the value 1.29 at  $1333^\circ\text{F}$  as recorded in table I, the constant activity of the two phases at this temperature being demonstrated in fig. 2.

## Method of Calculation, and Construction of Diagrams

In the past, long and—particularly to one unused to doing it—quite troublesome calculation has been required before one could predict, from the composition of the ingoing gas, what its reaction with the steel at temperature would be. The general procedure was as follows: First, select from table II values of the equilibrium constants appropriate to the temperature, and from them calculate, by a series of successive approximations, the composition of the gas mixture at equilibrium at temperature. Second, derive the carburizing (or decarburizing) power of this equilibrium mixture by evaluating its carbon activity  $a_c$  by means of the equations defining  $K_1$  or  $K_2$  (at equilibrium both give the same answer); comparison of this calculated  $a_c$  with the data in table I tells the carbon content of the steel with which this gas mixture is in equilibrium, and therefore shows whether it will carburize or decarburize a given carbon steel at that temperature. Third, calculate the ratio  $(\text{CO}_2)/(\text{CO})$  or  $(\text{H}_2\text{O})/(\text{H}_2)$  for the equilibrium mixture as a measure of its oxidizing potential toward the steel; comparison with the appropriate value of  $K_4$  or  $K_5$  in table II tells immediately if the gas tends to oxidize or reduce the steel.

The situation is inherently complex because, as we have seen, there are at equilibrium six composition variables, one for each of the gas constituents  $\text{CO}$ ,  $\text{CO}_2$ ,  $\text{CH}_4$ ,  $\text{H}_2$ ,  $\text{H}_2\text{O}$  and  $\text{N}_2$ , as well as the two additional variables of activity of carbon,  $a_c$ , and temperature, and because of the dependence of these variables upon each other through the equilibrium constants already discussed. Hence the description to follow cannot be made entirely simple and at the same time describe accurately the treatment used in combining the data and in converting the results to the final form. In what follows we have endeavored to present the treatment in as simple terms as are consistent with accuracy.

Description of the method of calculation, interpolation, and representation of the results will be divided into three sections corresponding to the three types of equilibrium considered: (1) equilibrium between the atmosphere and solid phases ( $\gamma$ —iron, iron + carbide, or graphite) with definite carbon activity, (2) equilibrium of the atmosphere with iron ( $\alpha$  or  $\gamma$ ) and its lowest oxide ( $\text{FeO}$  or  $\text{Fe}_3\text{O}_4$ ), (3) equilibrium with  $\text{Fe}_3\text{C}$  and iron oxide ( $\text{FeO}$  or  $\text{Fe}_3\text{O}_4$ ). At low temperature the fields of gas composition which are carburizing and oxidizing overlap. Investigation of this last equilibrium differentiates between atmospheres which are carburizing and those which are oxidizing.

The calculation in each section generally comprises three steps: determination of equilibrium gas composition at certain values of the variables carbon activity, temperature, etc., conversion of the results to the new composition variables, interpolation to rounded values of the variables and to other carbon activities and other temperatures, and finally representation in the form of graphs.

**Atmospheres with Definite Carbon Activity:** We start by pointing out that all of the variables men-



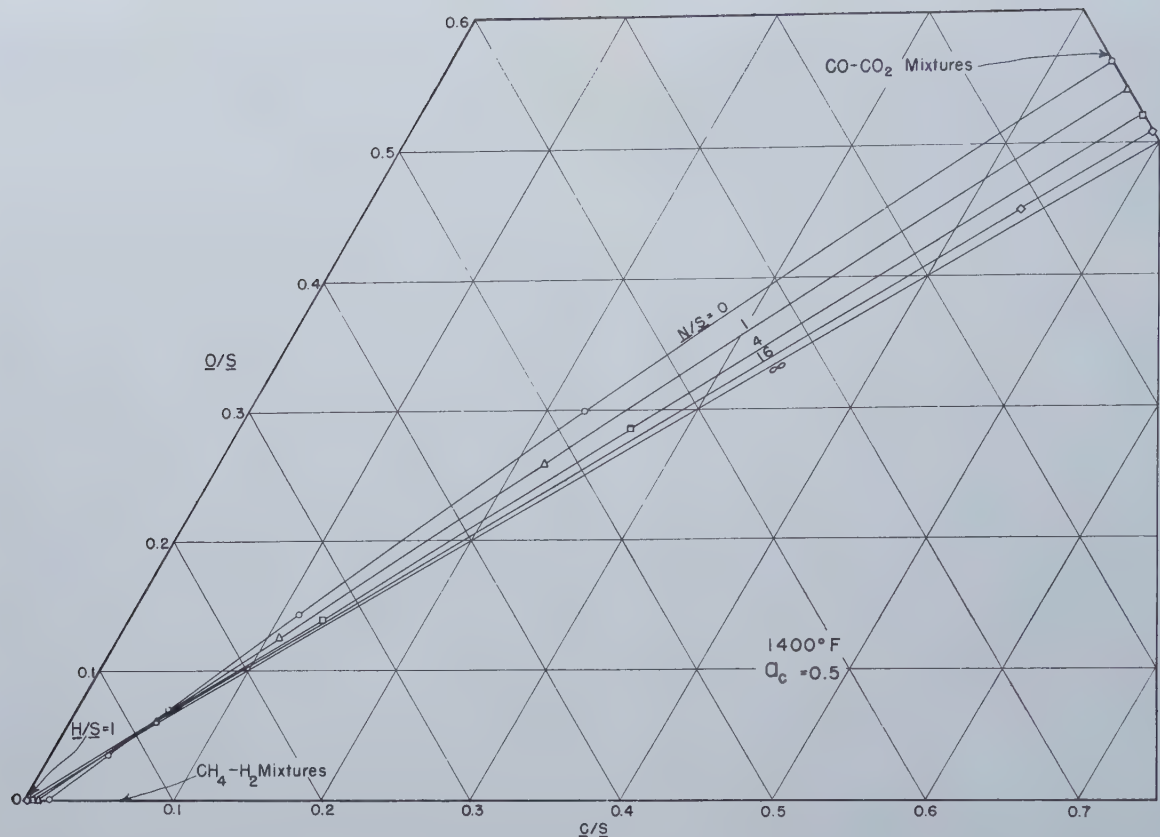


Fig. 3a (above)—Gas mixtures of several nitrogen contents ( $\text{N/S}$ ) values which at  $1400^\circ\text{F}$  have an activity of carbon ( $a_c$ ) of 0.5; triangular coordinate plot.

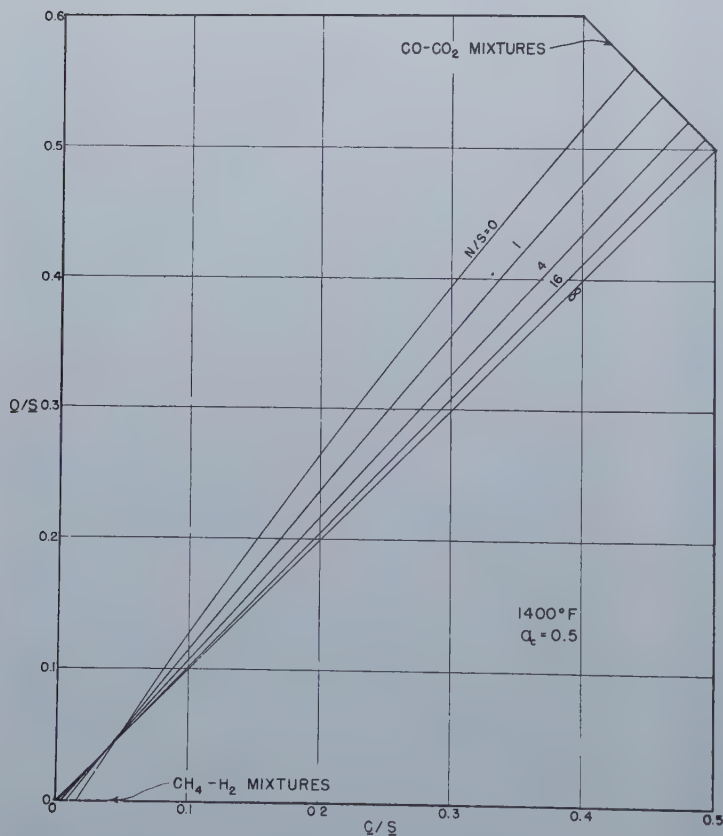


Fig. 3b (left)—Gas mixtures of several nitrogen contents which at  $1400^\circ\text{F}$  have a carbon activity of 0.5; rectangular coordinate plot.

tioned above are not independent. For example, if the temperature is fixed, arbitrary choice of three composition variables will fix the partial pressure of each gas constituent in equilibrium at temperature, provided we assume the total pressure is one atmosphere. Let us demonstrate this by choosing

values for  $(\text{CO})$ ,  $(\text{CO}_2)$  and  $(\text{H}_2)$ . Use of  $K_3$  immediately gives  $(\text{H}_2\text{O})$ ; combination of  $K_1$  and  $K_2$  yields  $(\text{CH}_4)$ ;  $(\text{N}_2)$  is the difference of the sum of all the others from unity; and  $a_c$  is derived from either  $K_1$  or  $K_2$ . In principle, choice of certain other independent variables serves equally well. In this

part of the work concerned with atmospheres of fixed carbon activity we chose (CO), (H<sub>2</sub>) and  $a_c$ ; the reason being that this gave an equilibrium composition with a certain chosen round value of  $a_c$ , rather than one with some unpredictable odd value. It should be noted that this general approach eliminates the need for successive approximations, but can be used advantageously only when a range of the variables is to be covered. Since we have to cover here a wide range of each constituent, we calculate equilibrium compositions at furnace temperature by selecting arbitrary values for certain of the variables, letting the values of the others fall where they may; later these equilibrium compositions will be transposed into the equivalent atomic compositions which also pertain to an atmosphere at room temperature going into the furnace.

To illustrate the procedure, if we set the temperature (thus fixing the equilibrium constants) and the carbon activity and select an arbitrary partial pressure of the two constituents (CO) and (H<sub>2</sub>), the molecular composition of the equilibrated gas is completely determined, the partial pressures of the other constituents being evaluated thus:

$$(CO_2) = \frac{(CO)^2}{a_c K_1}$$

$$(CH_4) = a_c K_2 (H_2)^2$$

$$(H_2O) = \frac{(CO_2)(H_2)}{(CO) K_3} = \frac{(CO)(H_2)}{a_c K_1 K_3}$$

$$\text{and } (N_2) = 1 - [(CO) + (CO_2) + (CH_4) + (H_2) + (H_2O)].$$

This relation merely expresses the stipulation that the total pressure shall be one atmosphere, as it nearly always is, within very close limits.

By performing such primary calculations for suitable increments of (H<sub>2</sub>) at a chosen value of (CO) we obtain a family of equilibrium "molecular partial pressures". Usually four or five well chosen increments of (H<sub>2</sub>) were sufficient to permit the required accuracy of interpolation at each value of (CO), and five or six well chosen (CO) values were sufficient to make graphs covering all gas compositions with the given activity of carbon at the given temperature.

The next step is the conversion of the "molecular partial pressures" to the new notation of "atomic partial pressure". This mode of representing the composition of the equilibrated gas at furnace temperature is particularly advantageous since it permits immediate comparison with room temperature (nonequilibrium) analysis. The new notation is based upon two facts: (1) in ordinary furnace atmospheres the reactive constituents CO, CO<sub>2</sub>, CH<sub>4</sub>, H<sub>2</sub>, H<sub>2</sub>O are all composed of the same three elements: carbon, oxygen and hydrogen; and (2) the composition of the equilibrated gas at any given temperature depends only on the relative amounts of these three kinds of atoms but not at all upon the molecular forms in which they were introduced into the furnace.

Consequently, the composition of the gas at equilibrium can be stated in terms of the proportions of the three reactive atoms—C, O, H—and of the inert gas nitrogen which is regarded as a dilu-

ent affecting only the partial pressure of the reactive constituents at a total pressure of one atmosphere. These "atomic partial pressures" we define as:—

$$\underline{C} = (CO) + (CO_2) + (CH_4)$$

$$\underline{O} = (CO) + 2(CO_2) + (H_2O)$$

$$\underline{H} = 2(H_2) + 2(H_2O) + 4(CH_4)$$

$$\underline{N} = 2[1 - (CO) - (CO_2) - (CH_4) - (H_2) - (H_2O)]$$

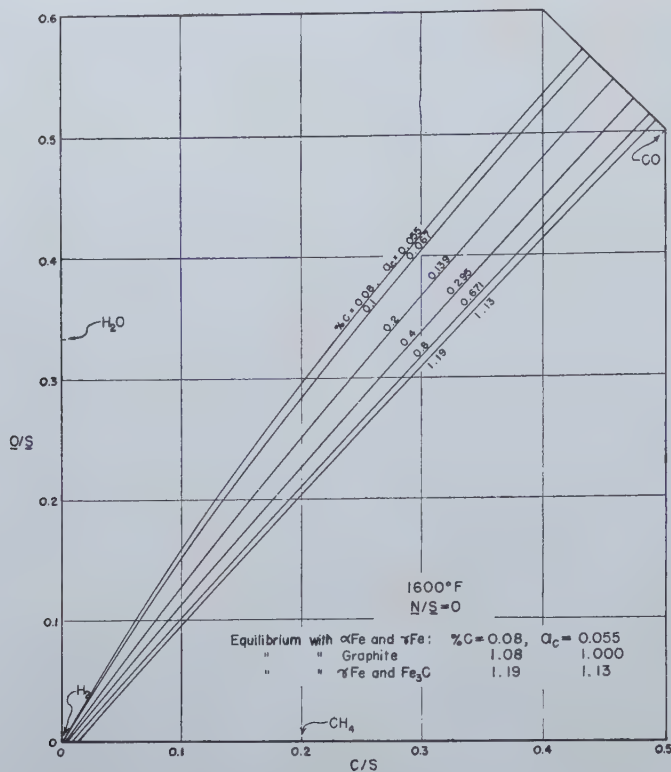
and use the symbol  $\underline{S}$  to designate  $\underline{C} + \underline{O} + \underline{H}$ , the sum of the active constituents. By this means the composition of the gas on the atomic basis is expressed in terms of the four quantities  $\underline{C}/\underline{S}$ ,  $\underline{O}/\underline{S}$ ,  $\underline{H}/\underline{S}$ , and  $\underline{N}/\underline{S}$ . The first three quantities represent the fraction of each active atom, and these three fractions always add up to unity so that  $\underline{H}/\underline{S} = 1 - \underline{C}/\underline{S} - \underline{O}/\underline{S}$ ; as will be seen later, this fact enables us to plot the three variables  $\underline{C}/\underline{S}$ ,  $\underline{O}/\underline{S}$ , and  $\underline{H}/\underline{S}$  on rectangular coordinate paper. The fourth quantity,  $\underline{N}/\underline{S}$ , which is commonly greater than unity, expresses the ratio of nitrogen to the total reactive atoms.

Since, for a reason which will be evident later, the final results are to be presented at regular increments of  $\underline{N}/\underline{S}$ , we now plot both  $\underline{C}/\underline{S}$  and  $\underline{O}/\underline{S}$  against  $\underline{N}/\underline{S}$  for each of the aforementioned families, and by interpolation at even increments of  $\underline{N}/\underline{S}$  get the corresponding values of  $\underline{C}/\underline{S}$  and  $\underline{O}/\underline{S}$ . The process is repeated for each family used in the primary calculations. The results enable us to construct a graph which covers all gas compositions with a given carbon activity at the given temperature; each line pertains to an even value of  $\underline{N}/\underline{S}$ . In the graph, fig. 3A,  $\underline{C}/\underline{S}$  is plotted against  $\underline{O}/\underline{S}$  and  $\underline{H}/\underline{S}$  on triangular coordinates. Since the sum of  $\underline{C}/\underline{S}$ ,  $\underline{O}/\underline{S}$  and  $\underline{H}/\underline{S}$  is unity we may equally well, and much more simply, plot  $\underline{C}/\underline{S}$  against  $\underline{O}/\underline{S}$  on rectangular coordinates. The result, fig. 3B, is the type of graph used in the final charts to depict gas composition. The value of  $\underline{H}/\underline{S}$  varies from unity, at  $\underline{C}/\underline{S} = \underline{O}/\underline{S} = 0$ , to zero at  $\underline{C}/\underline{S} + \underline{O}/\underline{S} = 1$ ; the latter condition is represented by the line designated "CO-CO<sub>2</sub> Mixtures".

On this graph only carbon, oxygen and hydrogen content are plotted directly and nitrogen content (or change in pressure of the three reactive elements) is shown by a series of lines, always labelled as to  $\underline{N}/\underline{S}$  value. The value of  $\underline{N}/\underline{S}$  is zero when the gas is composed entirely of the elements carbon, oxygen and hydrogen, infinity when  $\underline{C} + \underline{H} + \underline{O}$  approaches zero. It will be noted that the addition of nitrogen to a gas mixture originally nitrogen free does not change the position on the graph of the point representing the composition (since  $\underline{N}/\underline{S}$  is not plotted directly but only alters the position of the line against which the point is compared to ascertain whether the given gas has higher or lower activity of carbon.

**Interpolation at Intermediate Values of Carbon Activity (at Constant Temperature):** By the method indicated above, a series of curves on a graph was obtained which represents the composition of all atmospheres, at a temperature, which have a certain activity of carbon. Such a graph was constructed at each of four values of  $a_c$  at each temperature. Interpolation at intermediate steps of  $a_c$  (it being impracticable to make calculations for every small increment of the variables) was made by plotting  $\underline{C}/\underline{S}$  at constant  $\underline{O}/\underline{S}$  and  $\underline{N}/\underline{S}$  against the logarithm of  $a_c$ . From these curves (49 in number





at a given temperature)  $\underline{C/S}$  is read off at activities corresponding to selected increments of carbon content, the conversion from carbon percentage to activity being made by use of table I. Thus we now have the data necessary to construct a diagram, like those in the charts, comprising plots of  $\underline{C/S}$  against  $\underline{O/S}$  at constant  $\underline{N/S}$  (and constant temperature) for each of several selected carbon contents. Certain round percentages of carbon in  $\gamma$ —iron are included and also those in equilibrium with  $\alpha$ —iron,  $\text{Fe}_3\text{C}$  and graphite.

**Interpolation of Intermediate Temperature Levels:** Calculations as described above were made for 1400°, 1600° and 1800°F. Below the eutectoid, 1333°F, the extent of solution of carbon in  $\alpha$ —iron is so small that, for the present purpose, there is no interest in knowing how it varies with gas composition. Consequently, at 1200° and 1000°F calculations were made only for equilibrium with  $\alpha$ —iron— $\text{Fe}_3\text{C}$  and with graphite, and at 800°F for equilibrium with graphite alone, since at 800°F carbon diffuses so slowly that carburization or decarburization to any appreciable depth becomes impossible. The 800°F equilibrium with graphite and also with  $\alpha$ —iron— $\text{Fe}_3\text{O}_4$ , described in the next section, were included in the calculations as an indication of the possibility of soot deposition or oxidation during the slow heating or cooling of box annealing.

As can already be seen, the calculations are laborious and so it was decided to obtain data for construction of final graphs at the intermediate temperatures of 1100°, 1300°, 1500° and 1700°F\* by

\* Charts at 800°, 1100°, 1300°, 1500° and 1700°F are not presented in this paper but are available upon request, as noted above.

suitable interpolation of the calculated results at the other temperatures. This interpolation was carried out as follows: The values of  $\underline{C/S}$ , already obtained at the series of temperatures and  $\underline{N/S}$  values, are plotted against temperature at constant  $\underline{O/S}$  and  $\underline{N/S}$  for each round carbon content and also for

Fig. 4—Nitrogen-free gas mixtures which at 1600°F are in equilibrium with each of several carbon contents covering the stable  $\gamma$  iron field.

equilibrium with  $\alpha$ —iron, graphite, and  $\text{Fe}_3\text{C}$ . The final graphs for the intermediate temperatures are obtained by reading from these curves.

**Atmospheres in Equilibrium with Iron and its Oxide:** Perhaps the most troublesome property of steel is its tendency to oxidize. Scaling during heat treatment is common and usually detrimental. Consequently, it is desirable to consider the limitations imposed by equilibrium of the gas mixtures with iron and its oxide.

As in the case of equilibrium at a given carbon activity, calculation is performed for a series of chosen ( $\text{H}_2$ ) values at each of several round increments of ( $\text{CO}$ ). The pressures of the other constituents are obtained by use of the following relations from table II.

$$(\text{CO}_2) = K_4 (\text{CO})$$

$$(\text{H}_2\text{O}) = K_5 (\text{H}_2)$$

$$(\text{CH}_4) = \frac{K_2 (\text{CO})^2 (\text{H}_2)^2}{K_1 (\text{CO}_2)} = \frac{K_2}{K_1 K_4} (\text{CO}) (\text{H}_2)^2$$

The equilibrium pressures of the six molecular constituents are converted, as before, into the atomic pressure fractions  $\underline{C/S}$ ,  $\underline{O/S}$ ,  $\underline{H/S}$ ,  $\underline{N/S}$ .  $\underline{C/S}$  and  $\underline{O/S}$  for a given ( $\text{CO}$ ) family are plotted, just as

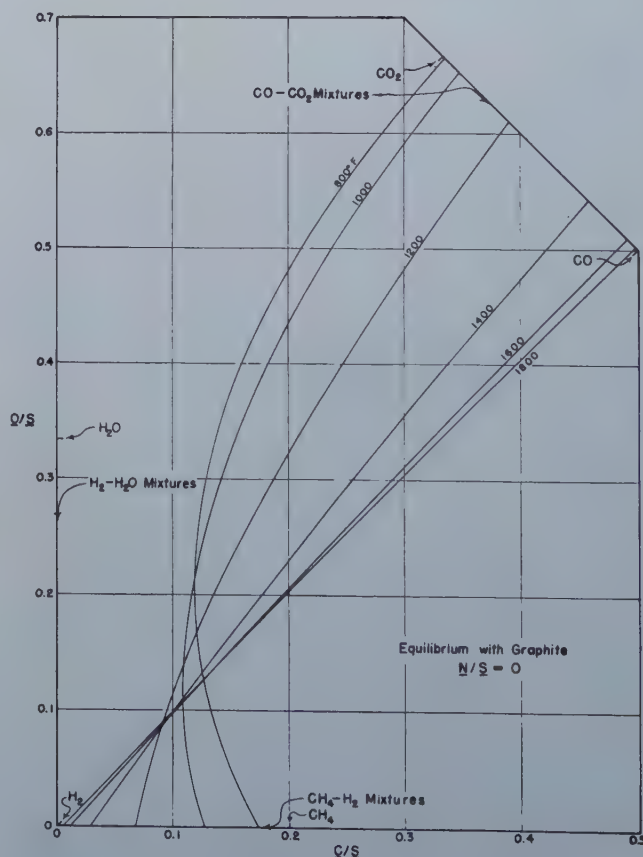


Fig. 5—Nitrogen-free gas mixtures in equilibrium with graphite at several temperatures.

before, against  $\underline{N/S}$ ; points at constant  $\underline{N/S}$  read off the curves for the several increments of  $\underline{(CO)}$  serve to determine the final curve of  $\underline{C/S}$  against  $\underline{O/S}$ . Thus we have at each temperature level a series of such curves, each at constant  $\underline{N/S}$ , completely analogous to the previous curves for a single carbon activity.

Similar treatment at each temperature of interest followed by interpolation similar to that described above, results in diagrams showing the gas composition in equilibrium with iron and its oxide over the whole temperature range.

**Atmospheres in Equilibrium with Iron Carbide and Iron Oxide:** Whenever the situation occurs that the gas mixture in carbon equilibrium with iron and  $\text{Fe}_3\text{C}$  is also oxidizing there arises the following uncertainty: in just what part of the overlapping region on the gas composition diagram does the gas tend to form  $\text{Fe}_3\text{C}$  and in what part iron oxide as the stable solid phase? Obviously both cannot be stable except along a line on the gas composition diagram which represents equilibrium between  $\text{Fe}_3\text{C}$  and iron oxide. This new calculation determines the gas mixture which is just able to oxidize  $\text{Fe}_3\text{C}$  and, unlike most other calculations in this report, does not involve iron as one of the phases in equilibrium. The general equilibrium reaction here considered may be obtained by combination of two simpler reactions as shown below. It involves a variable  $x$  resulting from the variation of the composition of the oxide, designated  $\text{FeO}_x$ , with temperature.

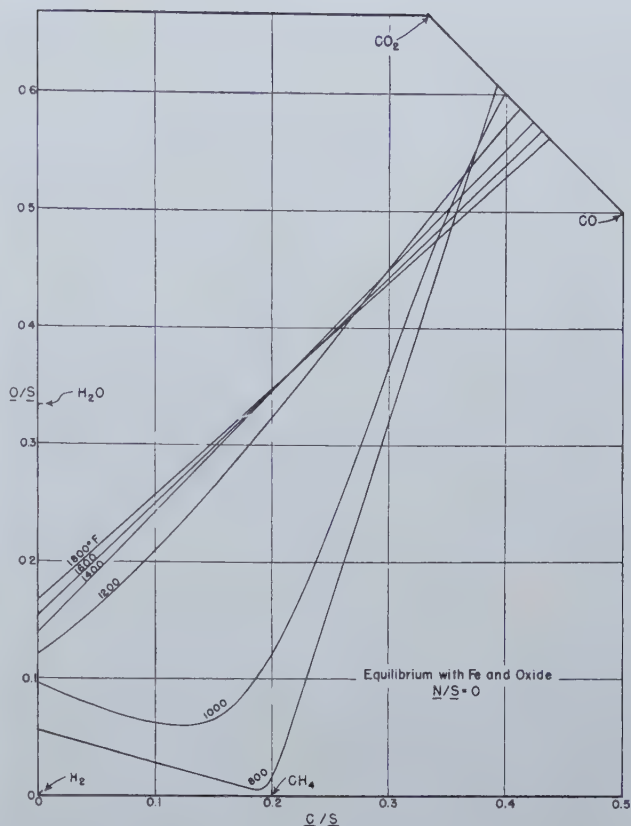
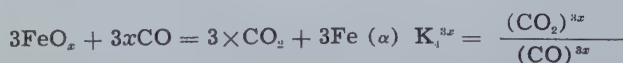
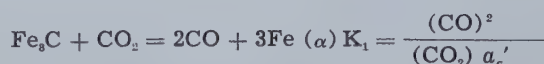


Fig. 7—Nitrogen-free gas mixtures in equilibrium with iron and its oxide at several temperatures.



Subtracting,



$$K = \frac{K_1}{K_1^{3x}} = \frac{(\text{CO})^{(2+3x)}}{(\text{CO}_2)^{(1+3x)} a_o'}$$

In these equations the activity of iron, of  $\text{Fe}_3\text{C}$ , and of  $\text{FeO}_x$  are all considered to be unity and  $a_c'$  is used to indicate that activity of carbon which is determined by equilibrium between  $\text{Fe}_3\text{C}$  and ferrite, no temperature being involved above the eutectoid.

Calculation of the equilibrium gas composition is made at each temperature for a series of increments of  $(\text{H}_2)$  at each of several chosen values of  $(\text{CO}_2)$ .

The following equations determine the partial pressure of the other gas constituents, the expression for  $(\text{CO})$  being derived from the equilibrium constant for the reaction above.

$$(\text{CO}) = \left( \frac{K_1 a_c'}{K_1^{3x}} \right)^{\frac{1}{2+3x}} (\text{CO}_2)^{\frac{1+3x}{2+3x}}$$

$$(\text{H}_2\text{O}) = \frac{(\text{CO}_2) (\text{H}_2)}{K_3 (\text{CO})}$$

$$(\text{CH}_4) = \frac{K_2}{K_1} \frac{(\text{CO})^2 (\text{H}_2)^2}{(\text{CO}_2)}$$

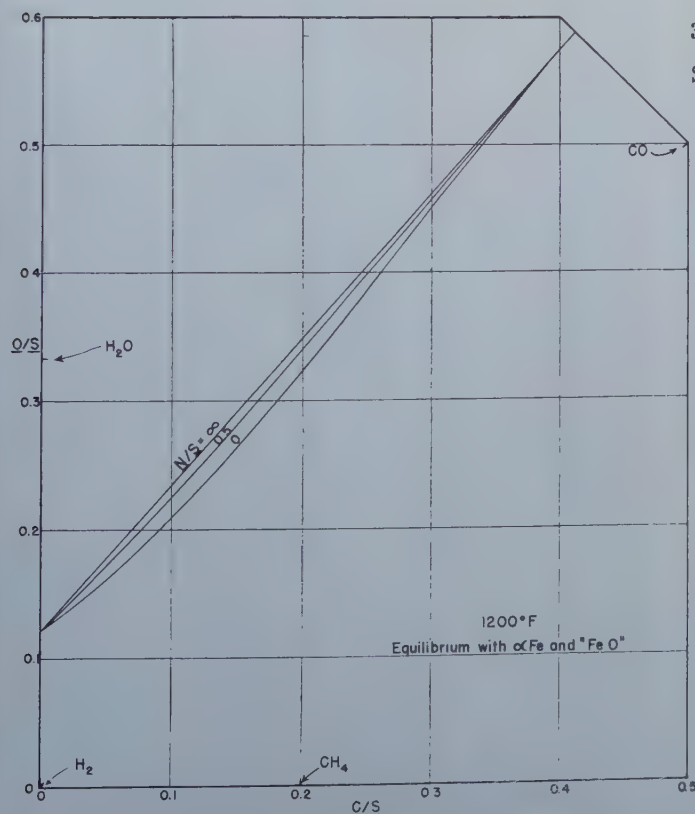


Fig. 6—Gas mixtures in equilibrium with iron and its oxide at 1200°F.



Values of  $x$  at 1000°, 1100°, and 1200°F were obtained from the composition of the oxide in equilibrium with iron<sup>7</sup> and are computed to be 1.333, 1.056, and 1.053 respectively at these temperatures.

The equilibrium molecular composition is converted to atomic fractions as before and  $\bar{C}/\bar{S}$  and  $\bar{O}/\bar{S}$  are plotted against  $\bar{N}/\bar{S}$  for each family at constant ( $\text{CO}_2$ ) in order to obtain  $\bar{C}/\bar{S}$  and  $\bar{O}/\bar{S}$  at the usual increments of  $\bar{N}/\bar{S}$  for the final plot. No further interpolation is needed since, in this case, calculations were made at each temperature for which the results are required.

The final results of all these calculations are presented in the form of working charts covering the range 800°–1800°F. All data at any one temperature appear on a single chart which is divided into segments, one for each value of  $\bar{N}/\bar{S}$ . Each segment has its individual  $\bar{C}/\bar{S}$  scale, indicated along the curves, but uses the common  $\bar{O}/\bar{S}$  scale shown at the left. This means of consolidating the results was adopted to facilitate their use.

### Discussion of Results

The calculation of equilibrium gas composition and the new type of graph used to express the results have already been described. A few remarks about this graph may help to explain just why it was used, and also enable the reader to use it more readily. The main purpose of this paper is to express gas equilibrium data in such a way that the effect of a gas mixture upon the charge at temperature can be estimated easily from a room-temperature gas analysis. To accomplish this not only must equilibrium calculations be made over the whole range of gas composition and temperature of interest but also the results must be expressed in a simple manner so they may be easily used.

The method used does simplify the expression of the equilibrium gas composition. We used three variables  $\bar{C}$ ,  $\bar{O}$  and  $\bar{H}$  to replace the five variables ( $\text{CO}$ ), ( $\text{CO}_2$ ), ( $\text{CH}_4$ ), ( $\text{H}_2$ ), ( $\text{H}_2\text{O}$ ) which otherwise would have to be shown. This is possible because the equilibria involved at temperature are concerned only with amounts of individual atoms, not with molecular forms in which they may exist at room temperature. In addition we divide each of these three by their sum so that the two variables  $\bar{C}/\bar{S}$  and  $\bar{O}/\bar{S}$  are all that are needed to express the five molecular partial pressures: this enables us to plot the equilibrium pressures of the five active molecules on an ordinary rectangular coordinate paper. Variation of the sixth constituent, nitrogen, is expressed by a series of curves, each at constant ratio,  $\bar{N}/\bar{S}$ , of atoms of nitrogen to the sum of the active atoms.

On the two dimensional graph of  $\bar{C}/\bar{S}$  vs.  $\bar{O}/\bar{S}$  the value of  $\bar{H}/\bar{S}$  at any point is always  $1 - \bar{C}/\bar{S} - \bar{O}/\bar{S}$ . Since nitrogen does not plot directly, we shall, in reference to gases in this paragraph, omit mention of it, keeping in mind, however, that any finite proportion of nitrogen may be present at any point without altering the veracity of the statements. Pure molecular hydrogen on the graph is plotted at  $\bar{C}/\bar{S} = \bar{O}/\bar{S} = 0$ ; pure  $\text{CH}_4$  is plotted at  $\bar{C}/\bar{S} = 0.2$ ,  $\bar{O}/\bar{S} = 0$ . All mixtures of  $\text{H}_2$  and  $\text{CH}_4$  fall on the straight line between these two points.  $\text{H}_2\text{O}$  is plotted at  $\bar{C}/\bar{S} = 0$ ,  $\bar{O}/\bar{S} = 0.333$ , and all mixtures

of  $\text{H}_2$  and  $\text{H}_2\text{O}$  are represented on the straight line between this point and the origin.  $\text{CO}$  is placed at  $\bar{C}/\bar{S} = \bar{O}/\bar{S} = 0.5$ ,  $\text{CO}_2$  at  $\bar{C}/\bar{S} = 0.333$ ,  $\bar{O}/\bar{S} = 0.667$ , and mixtures of these two on the straight line between them. Since along this line  $\bar{C}/\bar{S} + \bar{O}/\bar{S} = 1$ , this is a boundary of the graph. Compositions outside the triangle enclosed by this line and the two axes do not exist. Those outside the polygon defined by the points representing pure  $\text{H}_2\text{O}$ ,  $\text{H}_2$ ,  $\text{CH}_4$ ,  $\text{CO}$ , and  $\text{CO}_2$  are not of interest as furnace atmospheres. The field above the line between the points for  $\text{H}_2\text{O}$  and  $\text{CO}_2$  represents mixtures of  $\text{O}_2$  with these; that to the right of the line between  $\text{CH}_4$  and  $\text{CO}$  represents mixtures of these with higher hydrocarbons.

We have already discussed the calculation of the composition of all atmospheres which are in equilibrium with a given activity of carbon at a certain temperature. This information was given for  $a_c = 0.5$  at 1400°F in fig. 3B. As indicated on this graph  $\text{CH}_4 - \text{H}_2$  mixtures plot along the  $\bar{C}/\bar{S}$  axis ( $\bar{O}/\bar{S} = 0$ ), and  $\text{CO} - \text{CO}_2$  mixtures fall along a line such that  $\bar{C}/\bar{S} + \bar{O}/\bar{S} = 1$ . At a given value of  $\bar{N}/\bar{S}$  all gas mixtures with the stated carbon activity fall along a single curve, starting with a certain  $\text{CH}_4 - \text{H}_2$  mixture and ending with a certain  $\text{CO} - \text{CO}_2$  mixture. As the proportion of nitrogen in the gas increases the ratio of ( $\text{CO}$ ) to ( $\text{CO}_2$ ) and of ( $\text{H}_2$ ) to ( $\text{CH}_4$ ) increases, and at infinite dilution with nitrogen,  $\bar{N}/\bar{S} = \infty$ , the active constituents are simply  $\text{CO}$  and  $\text{H}_2$ . The same is the case for any other activity of carbon and any other temperature. Thus we see that the one atmosphere which neither carburizes nor decarburizes any steel at any temperature is pure nitrogen and that this inert property is, from the practical standpoint, unaltered by the presence of small amounts of  $\text{CO}$  and  $\text{H}_2$ .

The interpolation described in a previous section enables us to represent all atmospheres at a given temperature and  $\bar{N}/\bar{S}$  value which are in equilibrium with  $\gamma$ —iron of several chosen carbon contents as a series of curves, as in fig. 4; this is the type of diagram used in the working charts. Curves are generally shown not only for certain round carbon contents but also for the boundaries of the stable  $\gamma$ —iron field (equilibrium with  $\alpha$ —iron and with  $\text{Fe}_3\text{C}$ ) and for equilibrium with graphite. As is expected, higher carbon content in the metal accompanies a larger ratio of ( $\text{CO}$ ) to ( $\text{CO}_2$ ) or of ( $\text{CH}_4$ ) to ( $\text{H}_2$ ) in the gas. Except in atmospheres high in hydrogen the variation in gas composition accompanying a given change in carbon content, say 0.1 pct, is much less at high than at low percentage of carbon. This results from the fundamental relation defining  $K_1$  (table II) and is associated with the fact that at a given temperature and  $\bar{N}/\bar{S}$  value a line connecting the points for  $\text{CH}_4$  and  $\text{CO}$  represents atmospheres with infinitely high carbon activity and the approach to this line with changing carbon content of the steel is therefore asymptotic.

For a given activity of carbon, say equilibrium with graphite ( $a_c = 1$ ), and a given  $\bar{N}/\bar{S}$ , say zero, a diagram, fig. 5, can be constructed showing the change of gas composition with temperature. It is well known that as temperature increases  $\text{CH}_4$  becomes less stable relative to carbon and hydrogen and that  $\text{CO}$  becomes more stable relative to carbon and  $\text{CO}_2$ ; these facts can be seen from inspec-



tion of the values for  $K_1$  and  $K_2$  in table II. This means that at higher temperature the ratio of  $(\text{CH}_4)$  to  $(\text{H}_2)$  will decrease, and that of  $(\text{CO})$  to  $(\text{CO}_2)$  will increase resulting in the positions of the curves shown. It is apparent from fig. 5 how an atmosphere containing considerable  $\text{H}_2$  can be carburizing at high temperature but decarburizing at low temperature, whereas other atmospheres can be decarburizing at high temperature and carburizing at low temperature. In the latter case soot may be deposited at low temperature, as in cooling of an annealing furnace. In the vicinity of unit carbon activity there is one atmosphere ( $\underline{\text{C}}/\underline{\text{S}} = \underline{\text{O}}/\underline{\text{S}} = 0.1$ ) whose carburizing tendency remains essentially constant with temperature. Inspection of the final charts permits the selection of certain other atmospheres which maintain roughly constant activity or carbon content over a certain range of temperature.

Thus it is seen that at high temperature the atmospheres with a given activity of carbon contain little  $\text{CO}_2$  or  $\text{CH}_4$  even when no nitrogen is present, ( $\underline{\text{N}}/\underline{\text{S}} = 0$ ), and the range of variation of proportion of active atoms with nitrogen content is therefore very restricted because of the proximity of the limiting line representing mixtures of  $\text{CO}$  and  $\text{H}_2$  which pertains to  $\underline{\text{N}}/\underline{\text{S}} = \infty$ . Likewise for a given  $\underline{\text{N}}/\underline{\text{S}}$  the area on the gas composition diagram corresponding to a given variation in percentage of carbon will be less at higher than at lower temperature; at high temperature the addition of a small percentage of  $\text{CO}_2$  to  $\text{CO}$  or of  $\text{CH}_4$  to  $\text{H}_2$  results in a greater change in activity of carbon. This decreased spread at high temperature in the curves representing different carbon contents at fixed  $\underline{\text{N}}/\underline{\text{S}}$  will be noticed in the working charts.

As shown in table II a fixed ratio of  $(\text{CO}_2)$  to  $(\text{CO})$  or  $(\text{H}_2\text{O})$  to  $(\text{H}_2)$  at a given temperature is in equilibrium with iron and its lowest stable oxide. Thus for either a simple  $\text{CO}-\text{CO}_2$  or  $\text{CH}_4-\text{H}_2$  mixture, a fixed point on the gas composition graph, fig. 6, represents this equilibrium regardless of the proportion of nitrogen present. For more complex mixtures, interaction of the different species of gas molecule perturbs this simplicity and in order to maintain the constant ratio of  $(\text{CO}_2)$  to  $(\text{CO})$  and of  $(\text{H}_2\text{O})$  to  $(\text{H}_2)$  in the equilibrated gas a different gross proportion of the active atoms is required for each different nitrogen content. The result at a given temperature is a band of curves each for fixed  $\underline{\text{N}}/\underline{\text{S}}$  which converges to a point at each end as shown. The curvature for zero nitrogen content is greatest, and rapidly diminishes with increasing nitrogen to the straight line joining the points for the two simple gas mixtures. At low temperature the width of the band is great but at higher temperature, above  $1300^\circ\text{F}$ , it collapses to the straight line. Fig. 7 shows the effect of temperature in changing the contour and the position of the curve representing nitrogen free gas mixtures in equilibrium with iron and its oxide. With increasing temperature the ratio  $(\text{H}_2\text{O})/(\text{H}_2)$  increases but  $(\text{CO}_2)/(\text{CO})$  decreases, a fact also shown in table II. This results in the curves crossing as shown; the point where several curves cross represents an atmosphere whose equilibrium relationship is insensitive to temperature. It can readily be seen from fig. 7 how an atmosphere can be reducing at high tem-

perature but very oxidizing at low temperature; this explains the formation of the commonly observed film of oxide produced during cooling of steel in an annealing furnace. At very low temperature the reducing requirements are great and the only salvation is the fact that here the reaction rate is very slow.

**Precautions:** This discussion would not be complete without some cautioning remarks regarding the application of the charts contained in this paper. It should be emphasized that the results are calculated for iron-carbon alloys and may be applied only to carbon steels, except as inferences may be derived for alloy steels on the basis of carbon activity. The charts represent the composition of a gas mixture at a total pressure of one atmosphere whose constituents are at the same time in homogeneous equilibrium with each other and in heterogeneous equilibrium with one or more solid phases. In practice under certain conditions either or both of these aspects of the complete equilibrium may not obtain, and in such a case the indications derived from the charts are not directly applicable. If a gas mixture is not in internal equilibrium, certain constituents, say the methane and hydrogen, might exist in such proportions that they tend to carburize a given steel; others, the carbon dioxide and carbon monoxide, might at the same instant tend to decarburize, while the gas mixture as a whole might be indicated on the chart as being in equilibrium.

At temperatures above about  $1000^\circ\text{F}$  equilibrium within the gas phase and also between the gas and solid phases is commonly established. Even here a finite time is required for homogeneous equilibration of the gas, so that the indicated results from the charts do not apply for a very fast rate of flow (in part because of the time required to heat the gas). It should perhaps be mentioned that a piece of steel held in an atmosphere indicated to be in equilibrium with steel of a given carbon content may upon analysis prove not to be of that carbon content, even at the surface, if the time at temperature is too short. When comparing the charts with furnace atmospheres, an effort should be made to obtain the composition of the gas in actual contact with the work; only this gas can react with the steel. Since the ingoing gas is often modified by leaks or scale in the furnace, air in the brickwork, moisture or oil on the work, etc., it is recommended that the exit gas be analyzed, or even better that probe samples be taken near the work.

## References

- <sup>1</sup> R. P. Smith: *Jnl. Am. Chem. Soc.* (1946) **68**, 1163.
- <sup>2</sup> Mehl and Wells: *Trans. Am. Soc. Metals* (1937) **25**, 429.
- <sup>3</sup> C. Wells: *Trans. Am. Soc. Metals* (1938) **26**, 289.
- <sup>4</sup> R. W. Gurry: *Trans. AIME* (1942) **150**, 147.
- <sup>5</sup> J. B. Austin and M. J. Day: *Jnl. Am. Soc. Metals, Symposium* (1941). L. S. Darken and R. W. Gurry: *Jnl. Am. Chem. Soc.* (1945) **67**, 1398. R. P. Smith: *Jnl. Am. Chem. Soc.* (1946) **68**, 1163.
- <sup>6</sup> Wagman, Kilpatrick, Taylor, Pitzer and Rossini: *Jnl. Research Nat. Bur. Stds.* (1945) **34**, 143.
- <sup>7</sup> *Jnl. Am. Chem. Soc.* (1945) **67**, 1398.
- <sup>8</sup> Schenck, Dingmann, Kirscht and Wesselcock: *Ztsch. anorg. Allgem. Chem.* (1929) **182**, 97.
- <sup>9</sup> *Jnl. Am. Chem. Soc.* (1930) **52**, 4268; (1933) **55**, 1376.
- <sup>10</sup> *Jnl. Am. Chem. Soc.* (1934) **56**, 1838.
- <sup>11</sup> *Jnl. Research, Nat. Bur. Stds.* (1939) **22**, 407.



# Thermal Expansion Characteristics of Stainless Steels

Between  $-300^{\circ}$  and  $1000^{\circ}\text{F}$

by D. E. Furman

The thermal coefficients of linear expansion for several stainless steels have been determined over the temperature range from  $-300^{\circ}$  to  $1000^{\circ}\text{F}$ . The steels studied include types 301, 304, 316, 347, 310 and 330. This wide selection of compositions allows an insight into the effects of austenite stability and alloy content on the expansion characteristics.

THE recent increased use of materials in low temperature applications has created a field for which the properties of austenitic iron, chromium, nickel alloys are particularly suited. Tensile and impact values of stainless steels at subzero temperatures have been made available in the technical literature but, as yet, little information has been published with respect to expansion characteristics. In order to supply this need, a study was made of several of the more common compositions. Those selected were types 301, 304, 316, 347, 310 and 330 stainless steel. Both the mean and instantaneous coefficients of linear expansion were determined over the temperature range from  $-300^{\circ}\text{F}$  to  $1000^{\circ}\text{F}$ .

## Procedure

The materials used were commercial grades with the exception of type 301 which was made in a 10# high frequency furnace. The compositions are listed in table I. All steels were given an anneal at  $1950^{\circ}\text{F}$

for 30 min and water quenched prior to machining the expansion specimens.

Expansion readings were taken on specimens 0.250 in. in diam by 4.000 in. long with a fused quartz tube expansion apparatus similar to that described by Hidnert and Souder.<sup>1</sup> Temperature measurements were made with a chromel-alumel thermocouple calibrated for use at subzero temperatures.

A bath of propane was used for temperatures from  $-300^{\circ}$  to  $-100^{\circ}\text{F}$  and one of alcohol from  $-100^{\circ}\text{F}$  to room temperature. These baths were contained in large mouthed Dewar flasks and cooled to the desired temperature levels by passing liquid nitrogen through glass cooling coils placed in the flasks. The baths were stirred constantly for temperature equalization and were heated by immersing metal rods which were replaced frequently enough to give a heating rate of approximately  $150^{\circ}\text{F}$  per hr. A resistance wound furnace was used for heating from room temperature to  $1000^{\circ}\text{F}$  and throughout this range rates of about  $450^{\circ}\text{F}$  per hr were maintained. The furnace was designed so that the temperature gradient over the specimen was within  $5^{\circ}\text{F}$ .

In most instances, the expansion readings were taken during heating from  $-300^{\circ}$  to  $+300^{\circ}\text{F}$  at

D. E. FURMAN is Metallurgist, International Nickel Co., Inc., Bayonne, N. J.

AIME Chicago Meeting, Oct. 1950.

TP 2825 E. Discussion (2 copies) may be sent to Transactions AIME before Nov. 1, 1950, and will be published May 1951. Manuscript received Nov. 3, 1949.

Table I. Compositions of Stainless Steels Tested

Type	Heat or Stock No.	C	Si	Mn	S	P	Ni	Cr	Others
301	T18131	0.13	0.54	0.80			7.25	16.91	
304	02298	0.068	0.53	0.65	0.007	0.024	8.49	19.19	
316	02398	0.057	0.58	1.77	0.012	0.026	12.70	17.78	2.38 Mo
347	02397	0.068	0.56	1.74	0.006	0.019	11.30	18.65	0.77 CB
310	02557	0.111	0.42	1.51	0.010	0.022	21.64	27.22	
330	02556	0.052	0.62	1.81	0.006	0.006	35.19	15.30	

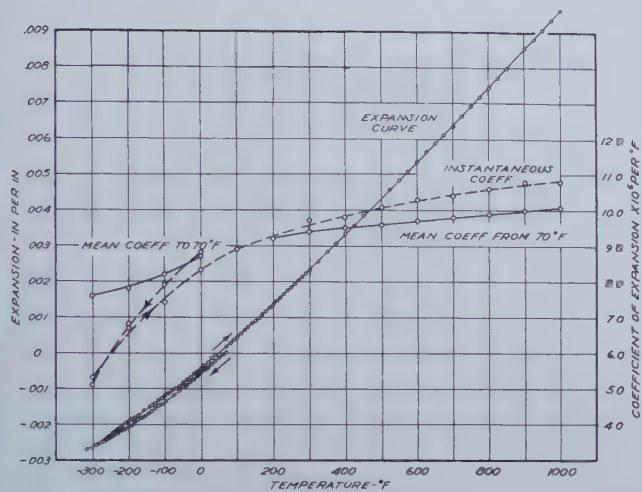


Fig. 1—Expansion characteristics Type 301 stainless steel annealed.

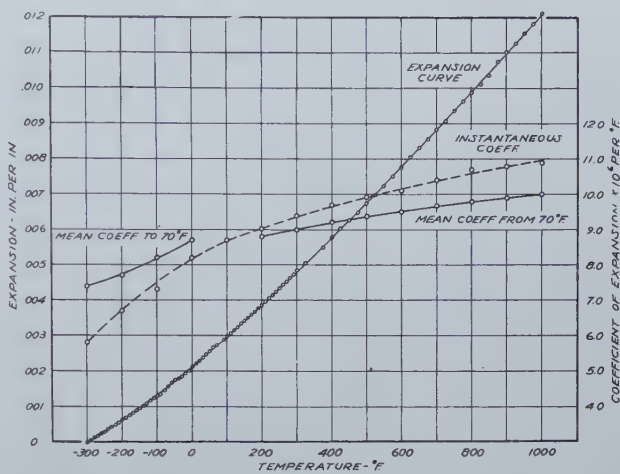


Fig. 2—Expansion characteristics Type 304 stainless steel annealed.

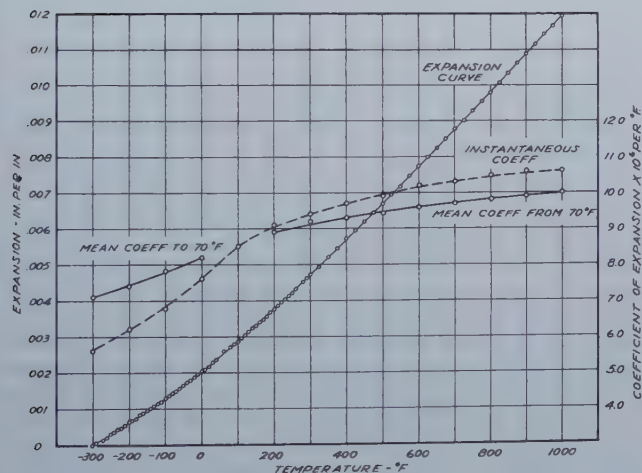


Fig. 3—Expansion characteristics Type 316 stainless steel annealed.

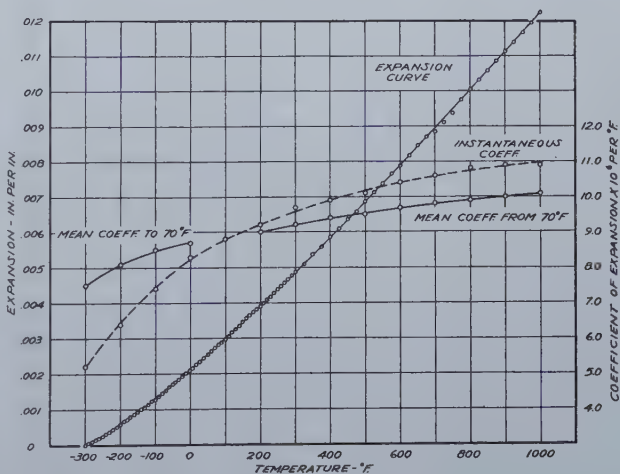


Fig. 4—Expansion characteristics Type 347 stainless steel annealed.

10°F intervals and from 300° to 1000°F at 25°F intervals. The readings were corrected for the linear expansion of the fused quartz specimen holder<sup>2</sup> and smooth curves were drawn through the plotted points ignoring small irregularities. The instantaneous coefficients were determined by tangents to the drawn curves at the indicated temperatures. In a few instances where duplicate specimens were run, good agreement was obtained.

#### Discussion of Results

The expansion curves for each of the stainless steels are plotted in fig. 1 to 6 together with the

thermal expansion coefficients. The instantaneous and mean expansion coefficients are also given in table II. For convenience in using the results obtained, the mean values were calculated with respect to 70°F, that is, room temperature.

The stainless steels studied covered a wide range of composition to give an insight into the effect of austenite stability as well as alloy content on the expansion characteristics. The least stable of the alloys investigated was type 301 and its behavior serves to show the influence of instability. In fig. 1, it will be noticed that a slight permanent expansion occurred after cooling to -320°F and reheating to



Ma- terial Type No.	Melt or Stock No.	—300°F	—200°F	—100°F	0°F	100°F	200°F	300°F	400°F	500°F	600°F	700°F	800°F	900°F	1000°F	Taken During Cooling from 70°F			
																9°F	—100°F	—200°F	—300°F
301	T18131	5.3	6.8	7.4	8.3	8.9	9.2	9.7	9.8	10.1	10.3	10.4	10.6	10.8	10.8	8.8			
304	02298	5.8	6.7	7.3	8.2	8.7	9.0	9.4	9.7	9.9	10.1	10.4	10.7	10.8	10.9		7.9	6.7	
316	02398	5.6	6.2	6.8	7.6	8.5	9.1	9.4	9.7	9.9	10.2	10.3	10.5	10.6	10.6				5.1
347	02397	5.2	6.4	7.4	8.3	8.8	9.2	9.7	9.9	10.1	10.4	10.6	10.8	10.9	10.9				
310	02557	4.9	5.9	7.1	7.7	8.2	8.5	8.9	9.3	9.4	9.6	9.8	10.0	10.2	10.3				
330	02556	3.1	4.7	5.9	7.1	7.7	8.3	8.7	9.1	9.3	9.6	9.8	10.0	10.2	10.4				

Ma- terial Type No.	Melt or Stock No.	$-300^{\circ}\text{F}$ to $70^{\circ}\text{F}$	$-200^{\circ}\text{F}$ to $70^{\circ}\text{F}$	$-100^{\circ}\text{F}$ to $70^{\circ}\text{F}$	$0^{\circ}\text{F}$ to $70^{\circ}\text{F}$	$70^{\circ}\text{F}$ to $200^{\circ}\text{F}$	$70^{\circ}\text{F}$ to $300^{\circ}\text{F}$	$70^{\circ}\text{F}$ to $400^{\circ}\text{F}$	$70^{\circ}\text{F}$ to $500^{\circ}\text{F}$	$70^{\circ}\text{F}$ to $600^{\circ}\text{F}$	$70^{\circ}\text{F}$ to $700^{\circ}\text{F}$	$70^{\circ}\text{F}$ to $800^{\circ}\text{F}$	$70^{\circ}\text{F}$ to $900^{\circ}\text{F}$	$70^{\circ}\text{F}$ to $1000^{\circ}\text{F}$
301	T18131	7.6	7.8	8.2	8.7	9.2	9.4	9.5	9.6	9.7	9.8	9.9	10.0	10.1
304	02298	7.4	7.7	8.2	8.7	8.8	9.0	9.2	9.4	9.5	9.6	9.8	9.9	10.0
316	02398	7.1	7.4	7.8	8.2	8.9	9.2	9.4	9.4	9.6	9.7	9.8	9.9	10.0
347	02397	7.5	8.1	8.5	8.7	9.0	9.2	9.4	9.5	9.7	9.8	9.9	10.0	10.1
310	02557	7.0	7.5	7.8	8.0	8.4	8.6	8.8	8.9	9.0	9.2	9.3	9.4	9.5
330	02556	5.8	6.5	7.2	7.6	8.1	8.3	8.5	8.7	8.9	9.0	9.1	9.3	9.4

Table II. Coefficients of Linear Expansion of Stainless Steels in Range  $-300^{\circ}$  to  $1000^{\circ}\text{F}$   
Mean Coefficients  $\times 10^6$  per  $^{\circ}\text{F}$

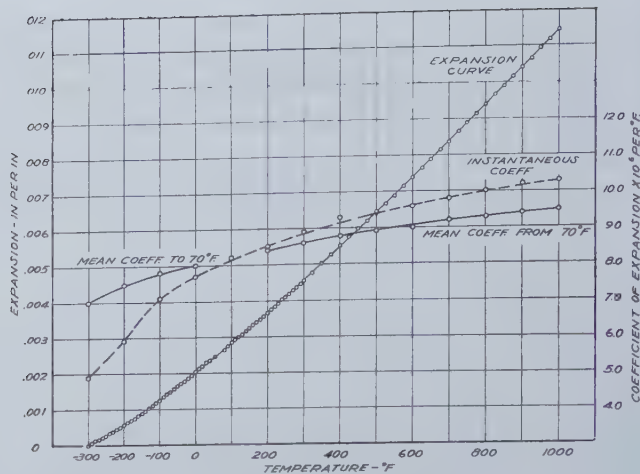


Fig. 5—Expansion characteristics Type 310 stainless steel annealed.

room temperature. This was attributed to the transformation of austenite to ferrite and magnetic measurements<sup>3</sup> indicated that the amount of ferrite formed was small, less than 2 pct. Evidently this quantity of ferrite was insufficient to affect the values of the instantaneous coefficients to any marked degree. As the instability of the annealed type 301 had a negligible effect on the expansion characteristics, there seemed to be little point in determining its effect on those of the more stable compositions. It should be remembered, however, that the presence of cold work can be expected to increase the instability of the marginal alloys and under favorable conditions might cause a sufficient amount of transformation to occur to lower the expansivity.

The effect of composition on the expansion characteristics at subatmospheric temperatures is similar to that at elevated temperatures. The austenitic stainless steels of high iron content tend to have higher thermal expansion coefficients than those with lower iron contents. As shown in fig. 7, types 301, 304 and 347 in the subzero range are quite similar and have the highest values of the steels studied,  $7.5 \times 10^{-6}$  per  $^{\circ}\text{F}$  at  $-300^{\circ}\text{F}$  to  $8.7 \times 10^{-6}$  per  $^{\circ}\text{F}$  at  $0^{\circ}\text{F}$ . This group continues to have comparatively higher coefficients from room temperature to  $1000^{\circ}\text{F}$  and in this range is joined by the molybdenum containing type 316. The type 310, which has a lower iron content, is little different from the type 316 at the subatmospheric temperatures but is definitely lower in the higher temperature ranges. A further reduction in the iron content, as represented by the type 330 stainless steel, results in the lowest values of mean coefficients obtained,  $5.8 \times 10^{-6}$  per  $^{\circ}\text{F}$  from  $-300^{\circ}$  to  $70^{\circ}\text{F}$  and  $9.4 \times 10^{-6}$  per  $^{\circ}\text{F}$  from  $70^{\circ}$  to  $1000^{\circ}\text{F}$ .

The reason for the very low expansivity of the type 330 at low temperatures is better understood if it is realized that this composition is essentially Invar containing 15 pct of chromium. As is known, this type of material goes through a reversible magnetic transformation or Curie point at a temperature which is dependent upon the alloy's composi-

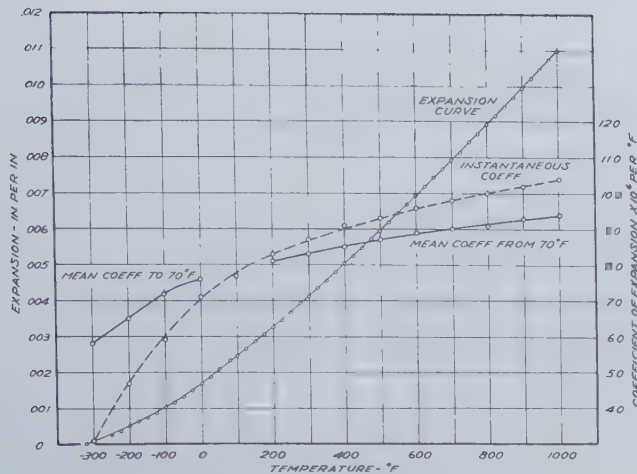


Fig. 6—Expansion characteristics Type 330 stainless steel annealed.

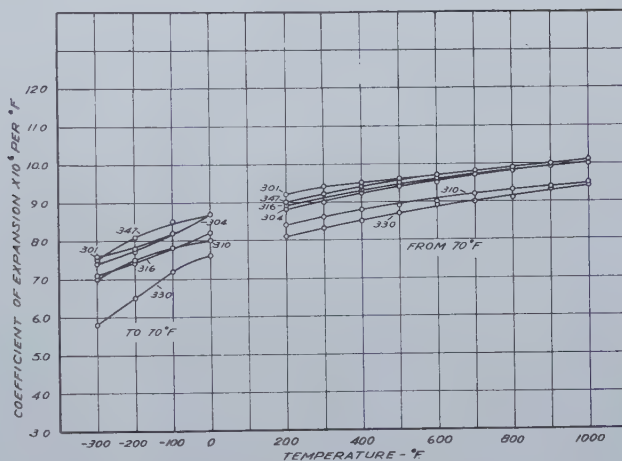


Fig. 7—Mean coefficient of expansion of stainless steels —300° to 1000°F.

tion. The transformation is accompanied by a sharp change in the slope of the expansion curve and the temperature at which it occurs is usually referred to as the inflection temperature. From the expansion curve in fig. 6, it is suggested that type 330 has an inflection temperature in the vicinity of  $-100^{\circ}\text{F}$  and this was checked approximately by a determination of the Curie point with a hand magnet. The existence of an inflection temperature is further illustrated by the rapid decrease in the instantaneous coefficient of expansion values at the low temperatures.

### Summary

A study of the expansion characteristics of commercial grades of chromium, nickel stainless steels in the annealed condition has been extended to temperatures down to  $-300^{\circ}\text{F}$ . The influence of austenite instability on the expansivities of the materials studied was found to be of small importance.

In general, it was found that the highest coefficients of expansion can be expected in those steels with the highest iron contents and any decrease in the iron content will tend to lower the

values. The comparatively higher iron alloys such as types 301, 304, 316, and 347 had mean coefficients of expansion of approximately  $7.4 \times 10^{-6}$  per  $^{\circ}\text{F}$  over the range  $-300^{\circ}\text{F}$  to room temperature whereas the corresponding values for the lower iron types 310 and 330 were  $7.0 \times 10^{-6}$  and  $5.8 \times 10^{-6}$  per  $^{\circ}\text{F}$  respectively. In the case of the type 330 stainless steel, the occurrence of a Curie or inflection temperature resulted in unusually low instantaneous coefficients of expansion below  $-100^{\circ}\text{F}$ .

### Acknowledgment

The author is grateful to N. B. Pilling, Director of Research of the International Nickel Co., for permission to publish these data and to V. N. Krivobok and W. T. Coughlin for their assistance.

### References

- <sup>1</sup> P. Hidnert and W. Souder: Dilatometry. ASM Metals Handbook—1948 Ed., p. 170.
- <sup>2</sup> International Critical Tables—Vol. IV, p. 21.
- <sup>3</sup> T. V. Simpkinson and M. J. Lavigne: Detection of Ferrite By Its Magnetism. *Metal Progress*, Feb. 1949, p. 164.



# Acid Conditioning of Metallurgical Smoke for Cottrell Precipitation

Early in the Cottrell application for recovery of metallurgical smokes, it was discovered that the presence of free acid in the precipitated dust was essential for efficient operation. This paper deals with the author's experience on the operation of an early Cottrell installation which led to the development of a sulphuric acid fuming furnace for auxiliary acid conditioning of combined Wedge roaster and lead sinter smoke.

by A. L. Labbe

SOON after the Cottrell treater was placed in operation in the Murray plant in 1918 to treat combined lead sinter and Wedge roaster smoke, it was noticed that the power flowing through the treater did not remain constant. This was indicated by the varying milliamperes and also by the total amount of power consumed by the rectifiers. At times, for then unknown reasons, the treater current fluctuated through a wide range of from 40 to 300 milliamperes. Fortunately, these variations in power did not affect the treater's overall recovery, as this installation consisted of three independent units in series, a feature which made the Murray Cottrell an outstanding installation over many years of operation.

Water conditioning of the smoke as a means of improving recovery was already known, but was not adaptable to this installation for reasons of excessive corrosion, which had been the case with other treaters using water as a conditioner exclusively. Tests conducted on the smoke had proved conclusively that water vapor played no part in the fluctuation of power, and the same was true of the  $\text{SO}_2$  contents and dust burden of the smoke. Neither did temperature variations of from 150 to 400°F have any effect on the power.

Finally it was noticed that the variation in power taken by the treater, and the efficiency of recovery, had some definite relation to the number of Wedge roasters operating, and particularly to the sulphur contents of the charge. This observation soon led to the discovery that free sulphuric acid was the conditioner for Murray smoke and that variations in acid contents of smoke accounted for fluctuations of treater power. Analysis of recovered dust revealed that only a few hundredths of a per cent free acid was necessary to maintain a very efficient recovery.

Once this knowledge was available, the roaster charge was adjusted as to sulphur contents to produce the necessary acid conditioner. For a number of years this practice was followed, but with improvements in the field of flotation, excess pyrites were eliminated from the smelting picture, so with a change in metallurgical practice we were confronted with the problem of providing the deficiency

in acid by some other means. The situation was aggravated and our problem of acid conditioning made more difficult by the increase in the lead contents of concentrates roasted resulting from better flotation methods.

The first step towards introducing acid vapors into the smoke stream by accessory means was accomplished by boiling sulphuric acid in cast iron pots placed in an open fire box. Acid fumes evaporated from the pots together with the combustion gases from the fire box were discharged into the flue through a cast iron pipe.

Capacity of the iron pots was quite limited because of the comparatively slow rate of evaporation and the use of lump coal as a fuel, which did not lend itself to practical temperature control. This method of firing resulted in frequent pot failures due to overheating.

In spite of these incidental difficulties, encountered in any new venture, the pot evaporator demonstrated the practicability of fuming sufficient acid to make up the deficiency in that naturally evolved in the Wedge furnace operation.

As time progressed, less and less high sulphur material was available for Wedge roasting, and the necessity for accessory acid conditioning reached a climax when the supply of this material was entirely eliminated.

Since the cast iron pots could only evaporate a few hundred pounds of acid per day, and were costly to replace, the logical thought presented itself of spraying the acid in a heated chamber. This new type of acid evaporator was simpler and more economical to operate and had sufficient capacity to fume several thousand pounds per day. The attached fig. 1 presents the outstanding features of the new furnace. A is the vaporizing drum, which is heated by the combustion gases of coal stoker fire box, B. Acid is sprayed into the hot gas stream within the drum through a number of air-acid atomizers, C, and the acid vapor, together with combustion gases, is introduced into the smoke stream through two cast iron pipes extending to different points in the flue.

The operation of this type of furnace requires control of temperature inside the drum to prevent overheating, which dissociates the acid vapor to  $\text{SO}_3$  and eventually to  $\text{SO}_2$ , with resulting loss of acid. The dissociation to  $\text{SO}_2$  at elevated temperatures is comparatively high, and in some cases accounts for as much as 60 pct of the acid sprayed into the drum. To reduce the loss of acid, through dissociation, some Cottrell plants, where the rate of acid sprayed reaches at times 9 ppm, use two evaporating units.

Fuming the acid into the smoke stream is only

A. L. LABBE is Consulting Metallurgist, American Smelting and Refining Co., Salt Lake City, Utah.

AIME New York Meeting, Feb. 1950.

TP 2824 D. Discussion (2 copies) may be sent to Transactions AIME before May 1, 1950, and will be published Nov. 1950. Manuscript received Oct. 17, 1949.





## Operation of A Basic-lined Surface-blown Hearth

For Steel Production

by C. E. Sims and F. L. Toy

**P**NEUMATIC processes for converting molten pig iron to steel were the major producers of steel during the latter half of the 19th Century and until shortly after the turn of the century, when these processes began losing ground to their rival—the basic open hearth process.

The first pneumatic process for converting molten pig iron to steel was introduced almost simultaneously by Sir Henry Bessemer in England and by William Kelley in the United States during the decade 1850-1860, but it was Bessemer who pushed it to a successful commercial operation. The process itself is so basically simple that it has not lent itself to many extensive or revolutionary improvements since its inception. During the first 30 yr, numerous<sup>1</sup> designs of vessel and many modifications of practice were tried before the industry standardized on something resembling current practice. The evolution in construction has resulted in proportions and arrangements based more on mechanical expedience than on metallurgical requirements.

Many of the early converters were stationary and had to be tapped similarly to the cupola or open hearth furnace. Such types were used for many years in Sweden and Germany, but the rotating or tilting types were favored in England and the United States. There was evident trouble with bottom tuyeres, because about half of the early designs were constructed for side blowing with tuyeres submerged below the metal-bath surface. Even portable tuyeres, introduced from above, were tried. The principal advance, which made bottom blowing practical, was the invention, in 1863, by Henry Bessemer, and subsequent improvement by A. L. Holley, of a detachable bottom so that the tuyere section could be renewed. In the meantime, it was

C. E. SIMS, Member AIME, is Assistant Director, Battelle Memorial Institute, Columbus, Ohio; F. L. TOY, Member AIME, is Assistant to the Manager, Research and Development, Carnegie-Illinois Steel Corporation, Pittsburgh, Pa.

AIME New York Meeting, Feb. 1950.

TP 2838 C. Discussion (2 copies) may be sent to Transactions AIME before May 1, 1950. Manuscript received Oct. 17, 1949.

the demonstration, by Robert Mushet, of the value of manganese in overcoming hot-shortness in converter steel, that made the process metallurgically feasible.

At first, only the acid practice was used, and Swedish pig iron, because of its low sulphur and phosphorus, was in great demand. The development of a suitable basic lining, by Sidney G. Thomas and Percy G. Gilchrist in 1876-1879, made the process applicable to the high-phosphorus ores of Europe. From the beginning of the use of the Thomas-Gilchrist process, trouble was experienced in obtaining sufficiently high temperatures in the blown metal, particularly in small heats. It is necessary, of course, to have the low-carbon blown metal at a considerably higher temperature for handling from vessel to ladle to molds than is needed for the molten pig iron.

All of the increase in temperature in a pneumatic process must be obtained from the heat of combustion of the elements contained in the pig iron plus that from silicon when ferrosilicon is added. In bottom blowing practice, carbon is almost completely burned, but inasmuch as it cannot be oxidized to a higher state of oxidation than to CO when in contact with liquid iron, only a minor portion of the heat-producing capacity is utilized. Under such conditions, it is not considered to be an efficient source of heat. In the acid practice, therefore, silicon and manganese are the large heat producers, are quickly oxidized, and are efficient fuels. Too much manganese, however, thins the slag greatly and tends to promote excessive slopping or throwing of slag and metal from the converter during the blow. Most operators, therefore, like to keep the manganese content of the iron below 0.60 pct, and thus the main source of heat is the silicon, which must be at least 1 pct and sometimes 2 to 3 pct for small blows such as for castings. Ferrosilicon is sometimes added during the blow to obtain an adequate temperature level.

In the basic (Thomas) practice, even more heat is required, because cold lime is added to flux the phosphorus after it is oxidized, and the basic slag must be maintained at a proper temperature. Silicon

A series of 1000-lb experimental heats of regular basic pig iron were surface blown with air jets on a basic hearth using burned lime to produce a basic slag. After a processing period of about 12 min, depending on the rate of air delivered, steels were produced with an average composition of 0.03 pct C, 0.08 pct Mn, <0.01 pct Si, 0.024 pct P, 0.021 pct S, and 0.003 pct N. Temperature was increased 300°F by the heat of reaction.

in the pig iron is undesirable because it requires the use of more lime. For that reason, it is kept down to about 0.6 pct. Manganese, relatively high as compared with acid practice requirements, is desired for its heat and may run as high as 1.0 pct. Phosphorus, however, is the main source of heat, and it is considered necessary to have at least 1.8 pct for good operation. In Europe, where the Thomas process is used extensively, most of the pig iron used contains approximately 2.0 pct of phosphorus. American basic pig iron, however, seldom exceeds 0.5 pct in phosphorus content, and for that reason, the Thomas process has not been used in America.

Even in the early days of bottom blowing, it was noted that when the vessel was tilted down to expose some of the tuyeres, a notable increase in temperature was obtained. This eventuated in the developing and patenting in 1891, by Tropenas, of an acid-lined converter for surface blowing with all of the tuyeres above the liquid level. The apparent reason for the long delay in this development was the firm conviction that the kind of violent agitation of the metal bath caused by blast air directed through it was essential to success. Tropenas proved that this was not true.

The Tropenas surface-blown converter produces much hotter blown metal from a similar charge than can the bottom-blown converter. Its ability to attain temperatures in excess of 3200°F (1760°C) when blowing cupola-melted iron has been demonstrated many times. As a consequence, it has been adopted almost exclusively for making small blows, particularly for steel castings. Being an acid process, however, it does not involve the method of adding cold materials as fluxes for slag making.

The reason surface blowing develops higher temperatures is generally attributed to the burning of CO, released by the carbon boil, to CO<sub>2</sub> above the bath, the heat release being within the vessel so that it is efficiently utilized in raising the temperature of the metal. Hall<sup>2</sup> disagrees with this and maintains that the extra temperature results from burning more iron. Certainly all of the CO is not burned to CO<sub>2</sub> in the vessel in this case, but much of it is, and the combustion of carbon to CO<sub>2</sub> produces 3½ times as much heat as when burned just to CO. On the other hand, more iron generally is burned during surface blowing.

Although there seems to be no record of a process which depends on surface blowing when conducted on a hearth composed of basic refractories using lime as a flux, it seemed possible to the authors that surface blowing under these conditions might furnish ample heat without the necessity of using high phosphorus pig iron. It will be remembered that basic pig iron, common to American practice, has not been considered suitable for use in a strictly pneumatic process, its phosphorus content of 0.2 to 0.4 pct being much too high for acid Bessemer practice and a great deal too low for the Thomas practice.

It is already known that when air is directed near the surface of liquid metal, a nitrogen content lower than the 0.010 to 0.018 pct normal to bottom-blown

converter steel results. However, it is generally accepted from experience in the basic Thomas process that the elimination of phosphorus and the realization of low nitrogen are incompatible in a pneumatic process. Therefore, when making steel of relatively low phosphorus and nitrogen content, it has heretofore been necessary to use the basic open hearth process. Although the basic open hearth process produces a steel of a quality which is in many respects superior to that of the acid Bessemer or Thomas processes and in large tonnages at competitive costs, it requires fuel from an external source. Moreover, it produces steel at a comparatively slow rate.

Thus it appeared desirable to investigate the possibility of producing by a pneumatic process low-nitrogen, low-phosphorus, open hearth quality steel from pig iron of a composition normally used in the basic open hearth process. Accordingly, Carnegie-Illinois Steel Corporation, with the benefit of the technical proposals of Battelle Memorial Institute, sponsored a cooperative project at Battelle Memorial Institute for this work.<sup>4</sup>

### Design and Construction of Experimental Furnace

An experimental unit of 1000 lb hearth capacity was decided upon. It was considered that this was about the smallest quantity of iron that could be blown to advantage, and the largest quantity that could be melted conveniently in the laboratory. A special high-frequency induction furnace was constructed for this work, and was powered by a 100-kva motor-generator set. This power is somewhat low for so large a quantity of metal and, to speed the melting operation, an aspirating gas burner was put in the top, through a hemispherical cover. Melting and superheating time was approximately 4 hr.

In designing the unit, efforts were made to keep it simple and yet as flexible as possible in order to obtain the maximum experimental data. The main dimensions and principal details of construction of the shell are shown in fig. 1. The shell was made in two parts, the nose or offtake section being separate. The lower section, or hearth chamber section, is a short cylinder with axis horizontal and a segment cut off at the top where the nose section is attached. The nose section is a truncated pyramid designed to carry away the gaseous products of reaction, in this case to a cupola stack, the 27-in. cupola having been removed to permit installation of the experimental unit.

There were several reasons for selecting this design. It gave a fairly shallow bath with a maximum depth of 9 in. and a large surface area approximately 18 by 36 in. The blast traveled the long dimension of the bath surface to give the maximum opportunity for reaction. It was thought that if this dimension were too short, misleading results might be obtained. Thus, it came close to simulating conditions that would pertain to larger hearths. By making a hearth chamber proportionately twice as wide, the capacity could be increased to over 27 tons by increasing the diameter three times.

The proper angle of the jets for best blowing con-



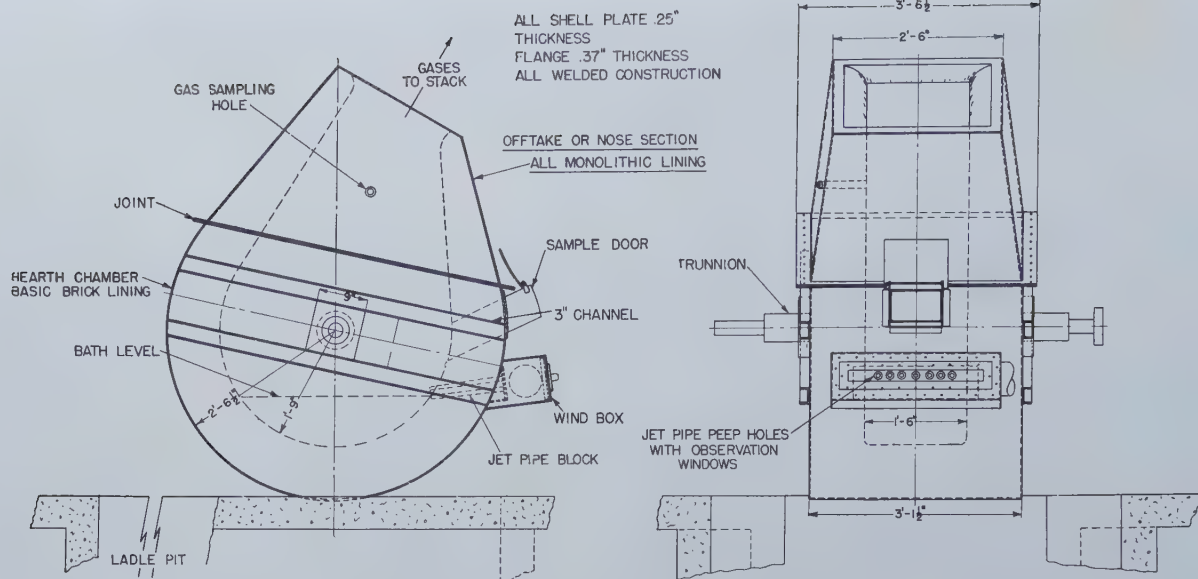


Fig. 1—Experimental hearth chamber for surface blowing.

ditions was not known, and this design made it possible to vary the angle from a minimum of  $7^\circ$  to a maximum of 20 to  $25^\circ$  without changing the shape of the bath.

The comparatively short vertical dimension and the rounded inside contour were designed to give a smooth sweep of the gases that would minimize turbulence and prevent excessive erosion of the refractory lining.

The design was made for simplicity in construction and lining with refractories.

**Lining:** The lower section or hearth chamber, with the exception of the jet-block area, was lined to a thickness of 9 in. with hard burned magnesite brick laid in magnesite mortar. Cardboard strips and ground furnace magnesite were used to provide approximately  $\frac{1}{8}$ -in. expansion per ft, and the bricks against the curved shell were backed up with an inch of ground magnesite next to the steel shell. Corrugated paper was first used between the side walls and the shell, but later this was replaced with ground high-heat duty insulating brick. A wedge-shaped wooden block was used at the location of the jet-pipe block during lining and was removed when the jet block was installed.

The nose section was made with rows of welded studs inside. An internal, removable form was inserted, and a periclase ramming mix was tamped into place to form a monolithic lining. This was 9 in. thick at the base and tapered to 5 in. at the nose opening. The two parts of the shell were lined separately and then assembled.

**Jet Pipes:** The jets consisted of seven pieces of standard 1-in. steel pipe each 13 in. long. These were placed in a single row  $2\frac{1}{4}$  in. between centers and threaded into a base plate like a row of candle molds. This manifold was placed in a form, and some of the same ramming mix used in the nose was rammed around the pipe to form a wedge-shaped jet-pipe block. The wooden block was removed from the lining through the wind-box and jet-pipe block inserted in its place. It was bolted down by means of the base plate to prevent leakage of air back of the lining.

The back plate of the wind-box was provided with a row of Pyrex windows arranged so that the

operator could sight down each pipe. The windows were mounted in screw caps which were easily and quickly removed during a blow if any pipe needed cleaning. These windows are shown in fig. 2.

**Mounting:** The complete unit was mounted over a pit large enough to allow it to turn over completely and also to contain a slag box and a ladle for receiving the finished metal. Tilting was done by hand through a large hand wheel and a worm gear. When the hearth and nose assembly was properly counterweighted, this was readily done. This is shown in fig. 3.

A protractor type of gauge and a pivoted, weighted pointer were mounted on the side. This was calibrated to give a direct reading of the jet-pipe angle for any position.

**Sampling:** Several methods were tried for sampling the bath during a heat, but the one found most satisfactory was to make an opening through the shell above the wind-box big enough, and at such an angle, that a regular sampling spoon could be dipped in and withdrawn with a sample of metal. Although this worked very well with metal samples, satisfactory slag samples were not obtained when the blast was on. In the first place, too little slag was obtained, and in the second place, frequent additions to the hearth left too much doubt as to the homogeneity of the slags. Samples of charge metal were taken before it was put into the hearth, and samples of finished metal and slag were obtained when the hearth was tilted for pouring. Almost every sample of metal and slag was analyzed by two laboratories, and in most cases excellent checks were obtained.

Gas samples were taken through a hole in the side wall of the nose section. The location was chosen because it was thought to be high enough to allow reactions within the hearth chamber to be fairly complete and low enough to avoid contamination from aspirated air. The gas samples were collected in standard glass sample tubes with stopcocks on both ends. One end of each sample tube was connected to a length of  $\frac{1}{8}$ -in. pipe by means of vacuum rubber tubing. The far end of the iron pipe was closed over with a piece of light-gauge sheet zinc,

**Fig. 2—Rear view of experimental furnace.**

Wind-box with peep holes, door for taking metal samples, hole for gas samples, and protractor gauge for adjusting angle of jet-pipes.



soldered in place. The whole tube was evacuated just before use, and the outer stopcock closed. When the closed end of the pipe was thrust into the gas sampling hole, the zinc melted in a matter of a few seconds and allowed the holder to fill with furnace gas. The inner stopcock was closed before taking the pipe from the gas stream to prevent the cooling and contracting gas from drawing in air.

**Temperature Readings:** Temperatures were determined with a Pyro Optical pyrometer. Readings on the pig iron were taken as it poured over the spout of the induction furnace or over the lip of the ladle into the hearth chamber. Finished-metal temperatures were taken as it poured over the lip of the nose section into the ladle. In these cases, the red scale with the standard correction for emissivity was used. Temperatures during the heat were taken by sighting through a window and down one pipe. Black body conditions apparently pertain here and the black scale was used. Excellent agreement was obtained between the two sets of temperatures. For example, the last temperature during blowing agreed with the temperature of the finished metal as it poured from the furnace.

**Auxiliary Equipment: Blast:** The blowing, recording, and controlling equipment installed for the cupola was available for use in delivering blast air. The Roots blower had a nominal capacity of 1200 cfm of air at 3-lb pressure. Within this capacity, the volume of air can be controlled and recorded automatically. Another instrument records the pressure. These are shown in fig. 3.

**Preheating:** Drying and low-temperature heating of the hearth and nose were accomplished by in-

serting an aspirating gas burner through the metal-sample door. This device was also used to keep the refractory lining from cooling down too much between heats. It could maintain a dull red heat for days.

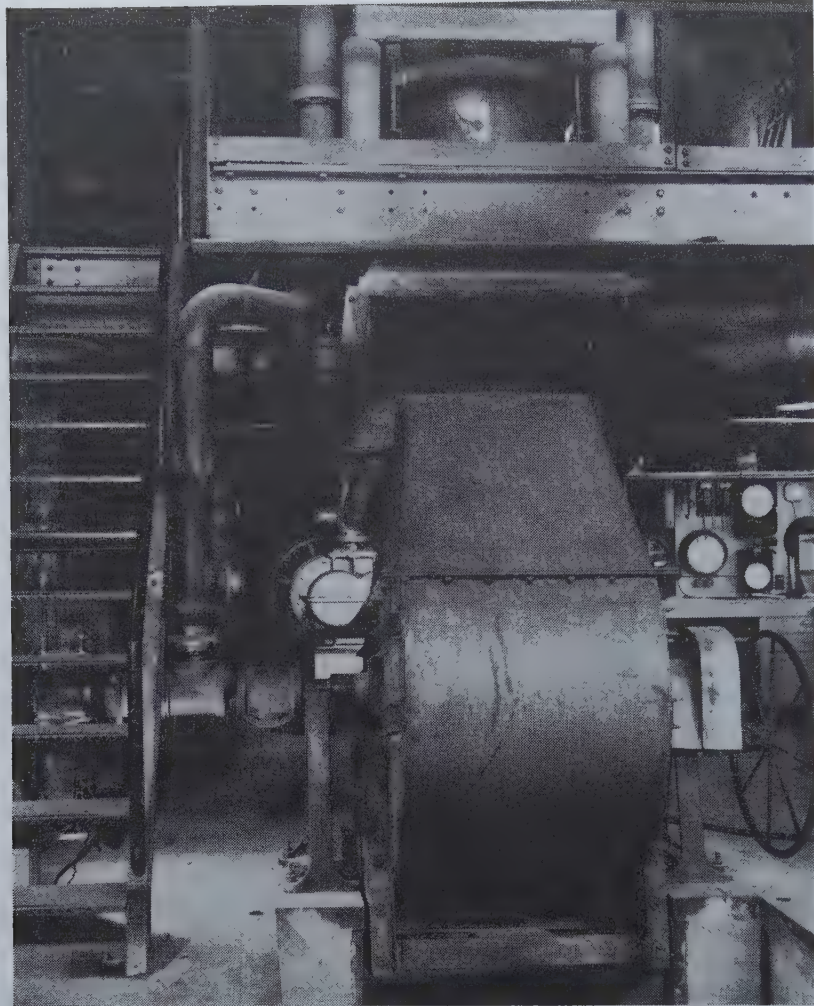
Inasmuch as only one heat per day could be made, it was necessary to preheat the lining and shell hot enough to simulate the temperature that would exist at the beginning of a heat during continuous operation. The jet-pipes were used as burners and natural gas as a fuel. At the beginning, a No. 3 Motor-Mix blower was connected into the air line through a T-connection. With this, the temperature of the lining could be raised to a maximum of 2500°F, which was not hot enough. This was remedied by tapping a high-pressure gas line directly into the air line from the Roots blower. After that, a pre-heat temperature of 2800°F was obtained readily. It was only necessary to shut off the gas to have the hearth ready for charging and blowing.

**Flame Record:** A photoelectric cell was arranged in a sighting tube and connected, through amplifiers, to a Speedomax recorder where changes in light intensity were recorded as voltage variations in a manner similar to that described by Work.<sup>3</sup> In practice, it was sighted from a distance of 30 ft on the flame where it issued from the nose. Although only a portion of the flame could be seen from this position, the cell proved to be extremely sensitive to variations in flame intensity.

### Experimental Work

Altogether, 32 heats were made, each with some variation from the others, to gather experimental data. Of these, three were brought to a premature





**Fig. 3—Front view of experimental furnace.**

Installation over pit, trunnions and tilting mechanism, counterweight, exhaust stack. Blower and control instruments in background.

end because of blocked jet-pipes and once a false indication of flame drop seemed to be obtained. Two consistent characteristics of the flame were noted early: first, there was no failure to obtain a "light" immediately when the blast was turned on, and second, the flame-drop was abrupt and definite when a carbon content of 0.04 to 0.05 pct was reached.

The first four heats were made under unfavorable circumstances. Besides the handicap of inexperience, both the molten pig charged to the hearth and the lining were too cold at the start. The first heat was stopped by plugged jet-pipes caused by dropping in 25 lb of pebble lime in a paper sack. The impact of this mass caused metal to surge into the jet-pipes where it froze. Also, when lime was added in this manner, it tended to fuse into one large mass which was not dissolved during the blowing time. In all later heats, the lime ( $\frac{1}{4}$ -in. pebbles) was poured in from a long-handled dipper.

The other three heats were finished successfully, although the temperature of the finished metal averaged 2860°F (1580°C), which is not hot enough for handling such a small quantity of very low-carbon steel. The third heat, which is summarized in table I, illustrates what could be done.

The silicon and phosphorus contents of the pig were high and the combustion of these gave considerable heat. Not only was the metal raised 500°F (280°C) in temperature, but the temperature of the lining was raised 200°F (110°C), and the 80 lb of additions were heated to bath temperature. In

spite of the high-silicon and phosphorus contents of the charge, over 97 pct of the phosphorus went into the slag.

After the first four heats, the lining was preheated to a temperature of approximately 2800°F

**Table I. Summary of Heat No. 3**

Analysis of, pct	C	Mn	Si	P	S	N
Pig iron	3.41	1.08	2.27	0.370	0.031	0.002
Finished metal	0.03	0.03	<0.01	0.007	0.024	0.003

	°F	°C
Temperature of lining before blowing	2860	(1450)
Temperature of metal entering hearth	2380	(1300)
Temperature of finished metal	2880	(1580)
Temperature increment of metal	500	(280)
Blowing time	15 min	
Burned lime added	75 lb	
Mill scale added	5 lb	

**Table II. Summary of Heat No. 10**

Analysis of, Pct	C	Mn	Si	P	S	N
Pig iron	3.77	1.80	1.61	0.444	0.039	0.008
Finished metal	0.04	0.04	0.02	0.019	0.023	0.004

Weight of molten pig iron to hearth	991 lb
Temperature of molten pig iron to hearth	2640°F (1450°C)
Weight of finished metal recovered	791 lb
Temperature of finished metal	2960°F (1630°C)
Weight of burned lime used	90 lb
Time to flame-drop	14 min
Afterblow	3 min
Total blowing time	17 min

(1540°C), and the pig was charged at an average temperature of about 2650°F (1450°C). This gave finishing temperatures ranging around 3000°F (1650°C) and greatly improved the operation. It is obvious, however, that the temperature of the finished metal did not increase as much as the starting temperature of the pig iron and the lining. Moreover, these heats showed that, in addition to ample heat being developed by the process and phosphorus being successfully eliminated, the latter was oxidized at the same time as the carbon instead of later as in the basic Thomas process. The steel so produced had a nitrogen content comparable to basic open hearth steel. These observations were confirmed in subsequent heats.

It was shown in Heat No. 3 (table I), that pig iron with a silicon content as high as 2.27 pct could readily be handled. In Heat No. 10 (table II), it was determined that phosphorus contents as high as 0.450 pct could be reduced to an acceptable level.

After the first ten heats, the practice was pretty well standardized, and a program was conducted to determine the effect of such variables as quantity of fluxes and time of adding, two slags vs. single slag, rate of air input, velocity of blast in jets, oxygen enrichment of the air, afterblow, and use of calcium ferrite. Methods of sampling were now worked out, and a photoelectric cell was set up to record variations in flame intensity. The standard conditions called for a preheat of the lining to about 2800°F (1540°C) and pig iron charged at a temperature of 2650°F (1455°C). Air was introduced at a rate of 750 cfm with a jet velocity of 300 fps and a wind-box pressure between 1½ and 3 psi.

A regular basic pig iron was used. After melting in the induction furnace it had a composition that averaged close to: C—3.95 pct, Mn—1.95 pct, Si—1.10 pct, P—0.31 pct, S—0.035 pct. The carbon was somewhat lower than in hot metal direct from a blast furnace, because kish was lost during freezing and remelting.

The metal analyses and heat data for 12 representative heats are shown in table III, and the flux additions and slag analyses for the same heats are listed in table IV. Inasmuch as this was all experimental work, most of these heats had one variable intentionally introduced while keeping all other conditions as nearly standard as possible. Details of six of these heats are shown graphically in fig. 4 to 9, inclusive.

A large number of gas samples were taken during the various heats and some of the results are shown in the figures for individual heats. On the whole, these samples were considered representative of conditions over the hearth, but it became apparent that individual samples were apt to be erratic and that no one sample was dependably representative of the conditions when it was taken. For

example, three samples taken simultaneously at different places in the same horizontal plane of the nose section gave the following results:

CO <sub>2</sub> , pct	CO, pct	O <sub>2</sub> , pct
15.4	11.7	nil
14.9	1.0	0.6
15.1	6.3	0.5

The presence of free oxygen in so many samples, particularly in the presence of CO\*, aroused the

\* There were no individual samples that contained a high content of both CO and O<sub>2</sub>.

Table III. Metal Analysis and Blowing Data for Twelve Representative Heats Showing Effects of Variables Intentionally Introduced

Heat No.	Sample	Metal Analysis, Pct						Temp. Iron Charged, °F	Temp. Hearth, °F	Temp. Finished Metal, °F	Pig Added, Lb	Finished Metal Recovered, Lb	Iron Recovered, Pct	Time to Flame-Drop, Min.	Total Blowing Time, Min.	Blowing Angle, Degrees	Remarks
		C	Mn	Si	P	S	N										
11	Pig charged Finished metal	3.84 0.04	1.61 0.06	0.84 0.02	0.319 0.023	0.027 0.003	0.005	2650	2780	2980	1013	758	81	13½	15½	11	2-min afterblow.
13	Pig charged Finished metal	3.80 0.04	1.59 0.11	0.95 0.01	0.333 0.017	0.028 0.025	0.003	2640	2740	2950	997	836	91	13	13	13	Standard heat.
15	Pig charged Finished metal	3.86 0.04	1.97 0.03	1.16 0.01	0.326 0.010	0.029 0.020	0.003	2610	2860	2910	1025	823	89	13	14	16-14	First slag removed after blowing 5 min. Some metal lost in operation.
19	Pig charged Finished metal	3.84 0.02	1.90 0.05	1.57 0.01	0.345 0.050	0.025 0.022	0.003	2660	2810	2960	1002	690	75	13	14	18	Two-slag operation. Blast volume reduced causing jets to plug.
20	Pig charged Finished metal	4.03 0.03	2.27 0.06	1.25 0.002	0.305 0.035	0.019 0.015	0.005	2620	2790	2900	993	832	90	10	10½	14	Excessive slopping near end.
22	Pig charged Finished metal	3.53 0.03	2.10 0.08	1.01 0.004	0.302 0.023	0.015 0.017	0.002	2650	2770	2970	998	822	89	12	14	14	High mill-scale addition.
23	Pig charged Finished metal	3.95 0.03	2.16 0.08	1.08 0.004	0.270 0.023	0.029 0.017	0.004	2650	2800	2940	1002	784	85	9	10	14	One-slag normal operation.
24	Pig charged Finished metal	3.97 0.02	2.15 0.08	1.12 0.01	0.302 0.028	0.030 0.018	0.003	2630	2750	2980	997	749	82	13	14	14	Blast volume increased 25 pct.
25	Pig charged Finished metal	3.81 0.03	2.14 0.07	1.05 0.01	0.290 0.033	0.031 0.025	0.003	2600	2700	3050	983	777	90	8	9½	14	Blast velocity increased to 430 fps.
27	Pig charged Finished metal	3.94 0.03	2.08 0.11	1.04 0.003	0.304 0.027	0.037 0.018	0.003	2630	2750	3170	934	694	75	14	18½	14	Air enriched to 25.6 pct O <sub>2</sub> .
28	Pig charged Finished metal	4.02 0.03	2.09 0.04	1.09 0.01	0.313 0.042	0.038 0.021	0.005	2600	2790	3200	997	797	88	12	12	10	Prolonged afterblow.
30	Pig charged Finished metal	3.93 0.03	2.05 0.05	1.04 0.005	0.301 0.013	0.041 0.018	0.004	2650	2700	2940	981	797	88	12	12	10	Calcium ferrite used as flux.



Table IV. Flux Additions and Slag Analyses for Same Twelve Heats Shown in Table III

Heat No.	Burned Lime, Lb			Mill Scale, Lb After				Spar, Lb	Slag	Slag Analysis, Pct								Remarks
	1st Slag	2nd Slag	Total	1st Slag	2nd Slag	Flame-Slag	Drop Total			CaO	MgO	MnO	SiO <sub>2</sub>	Al <sub>2</sub> O <sub>3</sub>	P <sub>2</sub> O <sub>5</sub>	Fe <sub>2</sub> O <sub>3</sub>	FeO	
11			60				10 10	2	Final	25.0	9.3	7.6	8.5	1.5	2.6	18.5	26.7	2-min afterblow.
13			60		10		10	2	Final	29.7	7.8	7.4	9.1	1.4	3.1	15.6	26.1	Standard heat.
15	10	35	45			5	5	2	Flush Final	10.6 17.3	18.4 17.4	16.5 5.4	17.8 6.2	1.0 0.9	2.8 2.0	11.6 18.9	21.5 32.8	Two slag. First slag off after blowing 5 min.
19	5	40	45			10	10	2	Flush Final	15.8 18.8	15.5 16.4	15.0 7.5	31.8 10.4	5.5 6.2	1.1 4.0	1.2 2.5	16.2 37.7	Two slag. Blast volume reduced causing jet stoppage.
20	5	40	45			15	15	2	Flush Final	13.1 13.9	17.3 16.0	17.8 5.9	28.8 6.2	6.4 3.3	1.6 2.2	1.3 6.7	16.5 52.6	Excessive slopping near end.
22	15	45	60			50	50		Flush Final	21.1 23.1	13.1 8.7	22.3 7.0	30.0 5.8	3.2 3.9	1.1 4.4	2.5 11.4	8.6 35.8	High mill-scale addition.
23			75		10		10 20		Final	24.0	5.9	8.8	8.9			8.0	40.1	Normal operation.
24			75		10		10 20		Final	20.1	13.5	8.3	7.6	0.9	1.7	17.1	31.3	Blast volume increased 25 pct.
25			75		10		10 20		Final	24.8	11.8	9.6	9.9			5.5	32.7	Blast velocity increased to 430 fps.
27			90		15		15		Final	23.6	16.7	8.9	11.1			7.7	26.7	Air enriched to 25.6 oxygen.
28			75				20 20		Final	13.4	17.7	6.1	8.0			5.1	45.2	4.5-min afterblow.
30			60 65*						Final	26.1	8.9	8.7	9.5			5.9	32.3	Calcium ferrite used as flux.

\* Calcium ferrite (CaO—47 pct, Fe<sub>2</sub>O<sub>3</sub>—41 pct)

suspicion that there was either an error in analysis or contamination with air. Careful checking proved that this was not the case, and forced the conclusion that representative samples could not be obtained at any one location. The high speed of the gases prevented uniform mixing and even complete reaction. Perhaps in a larger hearth chamber they would be more uniform. Any chance of using gas analysis as a control measure seemed to be precluded.

A sufficient number of specimens should give an accurate average, and in table V, the average results for 17 heats are given. To put all heats on an equal basis, they were divided into five periods, the four quarters of the time from start of blowing to flame-drop and the period after drop of flame. Analyses for samples in each period of any heat were first averaged and recorded opposite the heat number. The grand average is considered representative of a typical heat.

The average data are plotted in fig. 10. This figure, together with details for a normal heat such as in fig. 5, gives considerable information as to what goes on inside the hearth chamber. The line for total O<sub>2</sub> is obtained by calculating all oxygen-containing constituents back to O<sub>2</sub> by the formula

$$\frac{(O_2 + CO_2 + \frac{1}{2} CO) \times 100}{100 - \frac{1}{2} CO} = \text{equivalent } O_2.$$

**Progress of a Heat:** During the first 4 min of blowing, all of the silicon is oxidized and the manganese is just as rapidly depleted down to a stable content of between 0.2 and 0.4 pct. The carbon is oxidized slowly, being protected by the silicon and manganese. The phosphorus concentration actually increases. This is thought to be due, in part, to concentration from decreased bath weight and, in part, to phosphorus reversion from the lining and slag from the previous heat. Most of the sulphur drop occurs in this period.

There is a large increase in bath temperature during this period, because silicon and manganese

are good fuels. The flame intensity is only moderate at best and is very low during the third minute when almost no carbon is being burned. The intensity picks up as the carbon elimination is accelerated.

During this first period, the gas, as shown by fig. 10, contains moderate quantities of CO<sub>2</sub> and considerable O<sub>2</sub>, but no CO. The free oxygen indicates that nearly 1/3 of the air is not being used at all. In other words, the air is being used at about 70 pct efficiency. The deficiency in total oxygen indicates that nonvolatile oxides are being formed, that is, oxides of silicon and manganese mainly, but some of iron also.

As soon as the silicon is exhausted and the manganese reaches a stable content, the rate of carbon drop increases sharply and quickly reaches a rate of approximately 0.5 pct per min. This rate is maintained almost constant until the flame-drop when the carbon content is close to 0.05 pct. The phosphorus starts to drop simultaneously with the carbon (phosphorus is shown on an expanded scale). This is illustrated very well in fig. 4, 6, and 8.

During the silicon oxidation, there is very little boil, but immediately after the silicon is gone, there is usually a very violent boil which is almost explosive. This lasts not more than half a minute, after which the bath settles down to a steady but vigorous boil. The violent boil is nicely indicated by the peak in flame luminosity at 3½ min in Heat No. 24, fig. 6.

Bath temperature continues to increase up to about 8 min and then drops slightly toward the end of the blowing period. Part of this increase in temperature can be credited to the burning of phosphorus, but it is significant that the peak in temperature is reached when the content of CO<sub>2</sub> in the gas is at its maximum, as shown by comparison of fig. 5 and 6 with fig. 10. This observation seems to support the contention that it is the formation of CO<sub>2</sub> in contrast to the burning of carbon to a lower oxide (CO) in the bottom blowing processes

that gives surface blowing the advantage in temperature.

Fig. 10 shows that from the 4th to the 10th min of blowing practically all of the input of oxygen can be accounted for as oxides of carbon. This means that no non-volatile oxide such as iron oxide is being formed at this time. During this interval, only about 60 pct of the carbon is being burned to  $\text{CO}_2$  which would seem to indicate a deficiency of air. When air was put in at a greater rate as in Heat 24, fig. 6, however, the composition of the gas was not changed, the whole cycle was merely speeded up. Air delivered at two levels above the bath surface, originally proposed by Tropenas for his process, would, perhaps, burn a greater portion of the CO to  $\text{CO}_2$ .

The rate of carbon elimination is maintained unabated until a content of about 0.05 pct is reached and the flame intensity likewise shows no loss. When the carbon has reached its end point (it goes still lower during an afterblow), however, the flame drops abruptly, as though it had been turned off. It seems evident, therefore, that the flame intensity, at least as measured with the infrared sensitive photoelectric cell that was used, is a direct measure of the rate of carbon combustion. The eye sees what the photoelectric cell records but it is not so sensitive quantitatively. During the height of the heat, the flame is streaked with tiny incandescent particles, but just before the flame-drop, it takes on a soft phosphorescent luminosity that foretells the end. Every addition to the hearth caused a jog in the flame curve and mill scale produced a momentary flare up. This is shown by the interruption of the flame-drop in fig. 5.

After about 10 min blowing and when the carbon content has gone below about 1 pct, the content of CO and  $\text{CO}_2$  in the gas decreases rapidly and free oxygen begins to increase. The free  $\text{O}_2$  indicates a decrease of efficiency in the use of air. While air is cheap, a decrease in utilization means loss of heat units in the effluent gases. The rapid decrease in total oxygen at this time denotes the equally rapid formation of iron oxide. The manganese content of the bath is now decreased from about 0.3 to around 0.1 pct.

Although iron oxide is a desired constituent of the final slag, because of phosphorus elimination, it would seem desirable and economical to add it near the end of the heat as ore or mill scale rather than obtain it the hard and more expensive way (iron from ore at approximately 60 pct of the cost of iron from pig iron) by oxidation of the iron bath. Several attempts to increase the yield by this means were inconclusive.

It will be noted in fig. 10, that during the 6- to 9-min period, more oxygen is being given off than is being put in. This is believed due to the carbon reduction of iron oxide that formed earlier when there was a deficiency of oxygen in the effluent gas. Iron oxide added at this time should speed the carbon elimination. The reaction of added iron

Table V. Gas Analyses Averaged by Quarter Periods of Time to Flame-drop and Those Taken After the Drop of the Flame. Grand Average for 17 Heats Was Then Obtained

Heat No.	$\text{CO}_2$ , Pct					$\text{O}_2$ , Pct					CO, Pct				
	1	2	3	4	After Flame-Drop	1	2	3	4	After Flame-Drop	1	2	3	4	After Flame-Drop
11		10.1	16.1	16.2			1.6	1.6	2.0			0	7.9	2.6	
13		7.3	16.8	14.7	5.2		13.0	0.4	0.8	5.4		1.4	7.4	3.3	0.3
14		9.9		16.2	1.0		9.0		1.7	13.0		4.8		5.6	0.1
15		9.8			2.6		5.5			9.7		12.8			0
16	8.8	14.6	16.2	6.0		7.3	2.3	2.5	14.9		0.6	0	0.3	0	
17	7.6	16.2	14.5		0.7	8.7	0.8	0.3		18.8	0	0	16.0		0
19		8.8	16.5	5.3			11.3	2.3	10.3			1.7	8.1	0	
20	6.1	13.6	12.3	9.2		7.3	1.2	3.2	1.0						
21	8.1	8.6	14.4	13.6	1.6	5.5	1.4	0.1	0.5	8.1	0.1	24.0	11.1	17.0	0
22	7.0	14.2	15.8			7.7	1.9	2.0			0	9.9	2.7		
23	11.1	14.4	14.4	14.3	1.1	4.0	0.4	0.2	0	7.8	0.8	16.4	12.7	7.6	0.1
24	8.5	12.5	15.4	8.5		6.1	0.5	0.1	5.9		0	1.4	12.6	17.7	
25	8.6	14.4	14.5	9.8		6.7	0.1	0.2	1.8		0	10.5	14.1	8.9	
26	13.0	15.1	13.6		2.6	2.7	0.4	0		14.7	1.2	6.3	18.3		3.8
27	11.7	13.2	13.7	6.1		7.3	0.2	0.1	1.6		0	17.2	16.6	0	
28	8.8	14.1	14.2	11.7	0.2	5.2	0.2	0	0.9	3.5	0.2	5.5	13.5	17.9	0.5
29	6.6	6.6	15.4	12.3		7.8	7.3	0	0.3		0	0.1	11.0	23.0	
Avg.	8.8	12.0	14.9	11.1	1.9	6.4	3.4	0.9	3.2	10.1	0.2	7.0	10.9	8.6	0.6

oxide with carbon should be thermally self sustaining because so much of the carbon is eventually burned to  $\text{CO}_2$ . In Heat No. 22, tables III and IV, where mill scale was added at this period, there was a precipitate drop in carbon without loss of temperature. The mill-scale additions were later overdone, however, and the slag was chilled to the extent that it became inactive near the end of the heat and even caused a false indication at the flame-drop (final carbon, 0.26 pct).

**Afterblow:** After the flame-drop, the CO and  $\text{CO}_2$  contents of the gas drop rapidly toward zero, while the free oxygen increases rapidly. There is nothing left to burn now except iron, and the low total oxygen content shows that about half the oxygen of the blast is burning iron. The effect of afterblow is best illustrated in Heat No. 28, fig. 9, where it was continued for 4½ min. For the first 2 min of the afterblow, the bath temperature remained constant but then increased sharply and reached the highest temperature ( $3250^\circ\text{F}$  or  $1790^\circ\text{C}$ ) of any heat except No. 27 where oxygen-enriched air was used. This vindicates Hall's<sup>2</sup> contention that the burning of iron can furnish considerable heat. In the last 3 min before the flame-drop, when considerable iron is being burned, however, there is normally a slight drop in temperature. In Heat No. 28, the burning of iron during the afterblow caused an excessively low yield of 75 pct of the iron in the pig. For high yields, the blast should be stopped as soon as possible after the flame-drop.

**Phosphorus Elimination:** It is quite appropriate to compare this operation to the bottomblow basic or Thomas process. The normal change in composition of metal with blowing time in basic bottom blowing is shown in fig. 11. It will be noted that, while the elimination of silicon, manganese, and carbon is very similar in the two processes, the course of phosphorus elimination is significantly different. In basic bottom-blowing practice, no important quantity of phosphorus is removed up to the flame-drop and the afterblow is necessary for phosphorus removal. In the new process, on the other hand, all of the phosphorus that is going to be eliminated is out by the time the flame drops. In a number of heats where the blast was continued for 1 to 4.5 min after the flame-drop, no advantage in phosphorus removal was obtained.



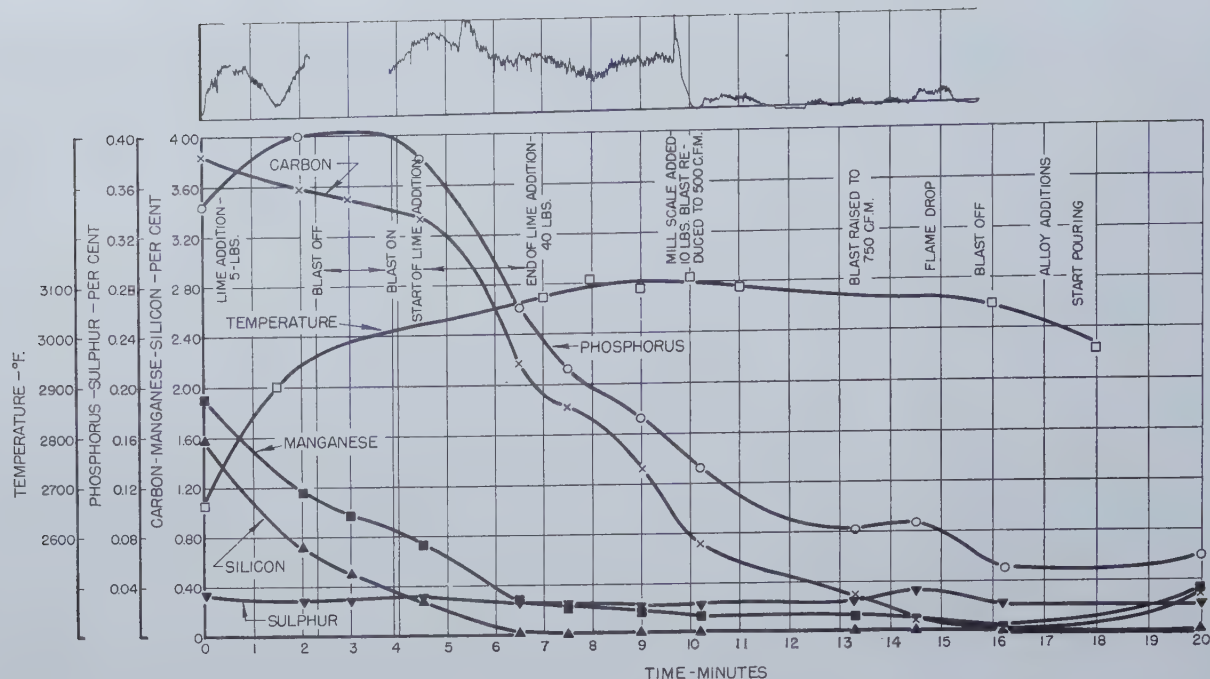


Fig. 4—Heat 19. Two-slag operation.

First slag removed after blowing 5 min. Blast volume reduced to 500 cfm at 9½ min.

In Heat No. 28, phosphorus reversion occurred during the afterblow, but this is attributed to the unfavorable temperature increase.

In bottom blowing converter practice, the air of the blast contacts the iron first and all of the oxygen is combined before it reaches the surface. Any oxygen the slag gets must be obtained from the iron, and that is not enough for phosphorus removal as long as any carbon remains. In surface blowing, the blast contacts the slag first and a strongly oxidizing slag is maintained throughout the heat. Indeed, it is probable that the bath obtains a substantial portion of the oxygen it uses from the slag instead of directly from the blast. That would explain why low blast pressures were so effective in these experiments.

Slag basicity is also necessary for phosphorus removal, and enough lime had to be added to leave a notable excess after part of it was neutralized by the silica. Most of it was added as quarter-inch burned lime. For the single-slag heats, a quantity of 60 to 90 lb or 120 to 180 lb per ton of pig iron seemed ample to lower the phosphorus to basic open-hearth ranges, provided the bath temperature was not unfavorably high. In tables III and IV, it may be seen that in Heat 13 an addition of 60 lb lowered the phosphorus from 0.333 to 0.017 pct. The slag on this heat contained 29.7 pct CaO and 41.7 pct  $\text{FeO} + \text{Fe}_2\text{O}_3$ . The temperature was normal at 2950°F (1620°C). Heat No. 27 had an addition of 90 lb of lime and yet finished at 0.027 pct phosphorus. The slag contained 23.6 CaO and 34.4 pct combined iron oxides. The relatively poor elimination of phosphorus in the latter heat is attributed to the abnormally high temperature of 3170°F (1740°C) due to the use of oxygen-enriched air. It should be mentioned that the high MgO content of these slags was caused by contamination from lining material. The bulk of this came from spalling rather than erosion and often unfused pieces were found in the slag. Obvious pieces were discarded in sampling the slag.

The MgO naturally contributed to the basicity of the slags and from that standpoint undoubtedly helped dephosphorization. In all of the single-slag heats, except those to which additions were made after the flame-drop, there was a marked tendency for the slag to end up with the following approximate contents:



In Heat No. 30, a burned-lime addition of 60 lb was made early, and then in the 9th min, 65 lb of calcium ferrite, containing 47 pct CaO and 41 pct  $\text{Fe}_2\text{O}_3$ , was added. Burned lime added at this time would not have gone into solution in the slag. The calcium ferrite blended with the earlier slag perfectly because of its low-melting point (about 2200°F or 1200°C). Although the opinion is held that it should have been added somewhat earlier for best effect, it did a gratifying job of dephosphorization.

**Two-slag Operation:** A number of heats were made in which a flush slag was taken off early in the heat to remove the bulk of the silica, and thus minimize the lime requirements. A little lime was added to the first slag to protect the lining, but most of it was added to form the second slag. It was found that the best time to stop the blast and take off the first slag was immediately after the silicon was all oxidized. It was taken off too early in Heat No. 19, fig. 4. If the carbon boil was allowed to get well under way before shutting off the blast and removing the first slag, it was difficult to reach the same rate of boil again and the temperature loss was not recovered. Heat No. 15 (tables III and IV) is a good example of what can be done in the two-slag practice. Phosphorus was lowered from 0.326 to 0.010 pct with the use of only 45 lb of burned lime.

The two-slag practice requires extra labor and extra time, while causing loss of sensible heat and loss of iron with the slag. It is questionable whether

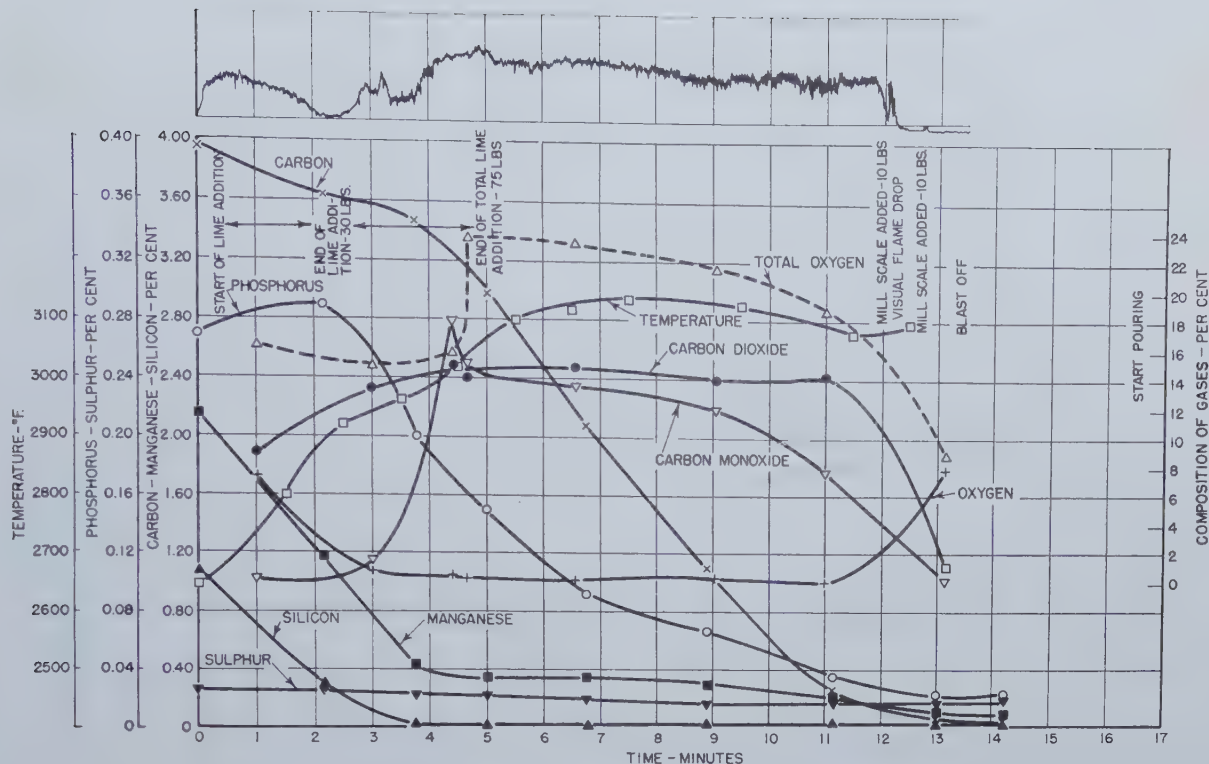


Fig. 5—Heat 23. Standard blowing conditions.

Mill scale added at flame-drop.

these disadvantages are offset by a saving of some 30 to 40 lb of burned lime per ton of pig iron used.

**Sulphur Elimination:** Although the strongly oxidizing conditions that exist during such an operation would not ordinarily be considered ideal for sulphur removal, it has come to be well known that a slag, if sufficiently basic, will remove sulphur from steel even under oxidizing conditions. In the present tests, the sulphur of the pig iron averaged 0.035 pct and that of the finished metal of all the heats averaged 0.021 pct, with a maximum of 0.028 pct and minimum of 0.011 pct.

Most of the sulphur eliminated was eliminated in the first 4 min of the heat, but there was usually a gradual reduction up to the end of the heats. High temperature seems to favor desulphurization, and the two high-temperature Heats, Nos. 27 and 28, had good sulphur elimination. On the other hand, high basicity without high temperature can give excellent desulphurization as shown by Heat No. 30.

**Slopping:** Slopping or violent ejection of slag from the mouth of a converter is a serious trouble often encountered in acid practice for both bottom-blowing and Tropenas-type converters and in bottom-blowing basic converters. Some bad slopping was encountered in the early heats of these experiments until it was discovered that the slopping could be prevented by maintaining a limey slag. In the two-slag practice, the first slag slopped badly if the carbon boil was allowed to start before the slag was removed. By adding lime early and maintaining a limey slag, trouble with slopping was absent until Heat No. 20 (see tables III and IV). During this heat, 15 lb of mill scale was added, but in addition, an excessive amount of iron was oxidized. As a result, the slag formed was low in lime, high in iron oxide, and very fluid. Slopping toward the end of the heat was very bad, large

"pancakes" of slag being ejected. After this heat, the low-lime high-iron oxide slags were avoided and there was no further trouble with slopping in the program at Battelle.

In subsequent, large-scale tests described in more detail later, other factors were noted which seemed to have an important influence on the tendency for slag ejection. A slag too thick and viscous was almost as bad as a slag too thin. When the jet velocity exceeded about 500 fps (340 mph), slopping was greatly intensified and it appeared that this was about the maximum jet velocity that could be tolerated.

One very important factor seemed to be the continuity of the carbon boil. When air is introduced at too slow a rate to promote a strong and continuous oxidation of carbon, the boil becomes intermittent, quiescent periods alternating with violent eruptions. During the quiet period, a high potential of iron oxide builds up, and the subsequent boil is almost explosive when it starts. Much slag is then ejected, often as exceptionally large "pancakes." A similar phenomenon occurred in some of the experimental heats but only at the very beginning of the carbon boil. This is illustrated by Heat No. 24, fig. 6. Near the end of the silicon oxidation, there was a relatively quiet period where the flame had low intensity. This was followed by a violent boil during which slag was ejected for about a half minute. After that, the boil was maintained at a steady, high level, and no more slag was thrown out. A steady boil, even though intense, keeps the slag in constant agitation, promotes proper temperature and viscosity, and prevents the formation of large gas pockets. A rough analogy would be a beaker of water which boils steadily when boiled rapidly but which will eject water when boiled too slowly.

**Blast:** A nominal blast volume of 750 cfm delivered for 12 min was found to be sufficient to blow



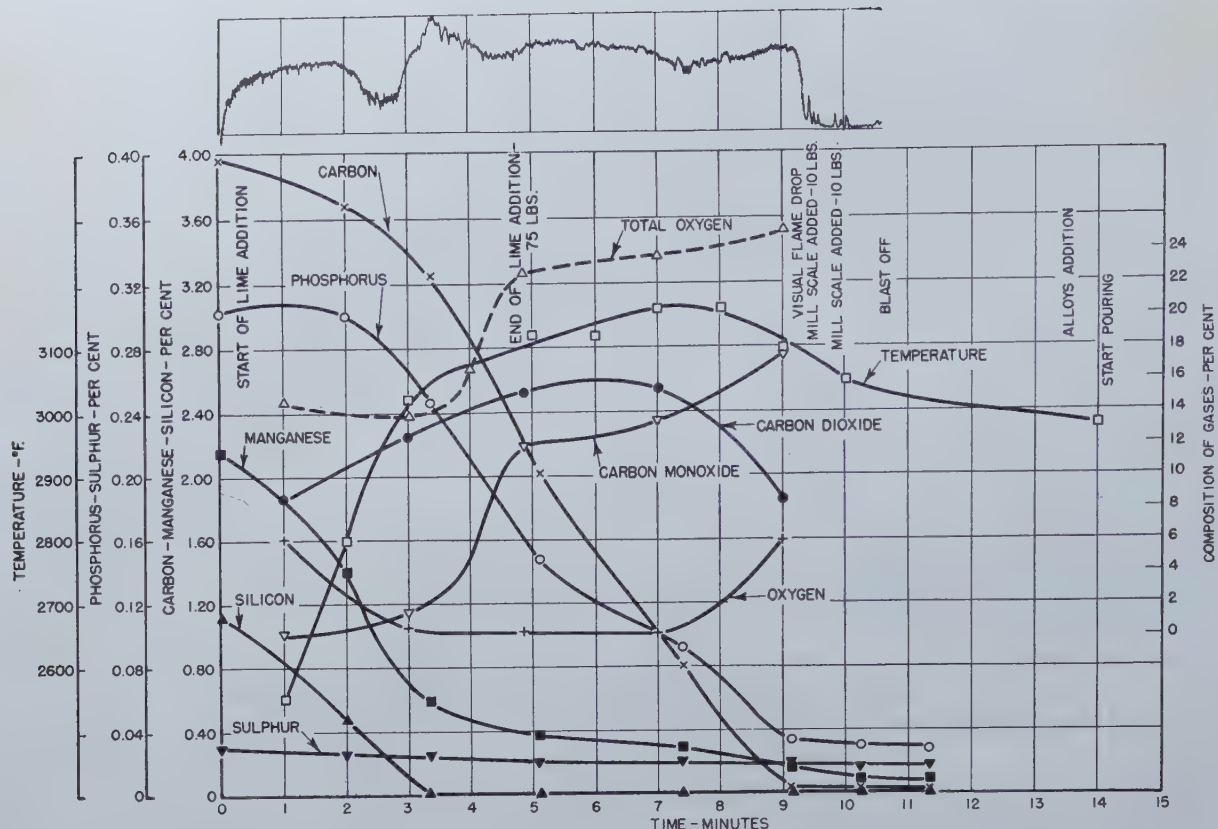


Fig. 6—Heat 24.

Blast rate increased about 25 pct, decreasing blowing time proportionally.

1000 lb of pig iron. This calculates to 18,000 cf per ton of pig. For most of the heats, this was delivered through seven 1-in.-diam jet-pipes. The pressure required to deliver this volume varied from about 25 oz at the start of the heat when the jet-pipes were clear, to 45 oz later in the heat when jet-pipes were partially blocked by slag which tended to chill around the exit ends. This slag was readily removed by thrusting a steel rod through the pipes.

A rate of 750 cfm through 1-in. jet-pipes gave a linear velocity of about 330 fps. This seemed to be close to the minimum velocity that would keep the pipes open, because when the blast volume was reduced to 500 cfm with a decrease in velocity to about 240 fps as in Heat No. 19, fig. 5, the pipes promptly plugged. Even though they were opened up and the blast volume later restored to 750 cfm, normal operation could not be regained.

In Heat No. 25, fig. 7, the 1-in. pipes were replaced with  $\frac{3}{8}$ -in. pipes and the pressure stepped up to deliver the same nominal volume of 750 cfm. The maximum pressure was 44 oz, but the jet velocity was increased to 430 fps. The time to drop of flame was 12 min 55 sec or 40 sec longer than Heat No. 23. A comparison of average flow rates and times indicated that almost exactly the same total amount of air was used in both heats. Except that the bath temperature of this heat was unusually high, the higher jet velocity seemed to have no effect.

The total amount of air required, or rather the total oxygen delivered by the air, was very consistent. When the blast rate was increased from 750 to 1000 cfm and the jet velocity to 440 fps, as in Heat No. 24, fig. 6, the whole cycle was speeded up and the time to drop of flame was lowered to just over 9 min. This reduction in time is inversely proportional to the rate of air input. The gas com-

position and other conditions of this heat were virtually unaltered.

In Heat No. 27, fig. 8, the oxygen content of the blast was increased to 25.6 pct by bleeding tank oxygen into the air delivery pipe. The blast rate of enriched air was maintained as close to 750 cfm as possible. Again, the cycle was speeded up with a flame-drop at 9 min. Thus, with the oxygen content increased by 25 pct, the time was shortened by 25 pct or in inverse proportion to the oxygen content. In this heat, the total oxygen content of the gases was higher than normal as might be expected, and at the peak of the heat, the CO content of the gas was definitely higher than the CO<sub>2</sub> content. The bath temperatures were very high, a maximum temperature of 3250°F (1790°C) being recorded near the end of the heat.

**Blowing Angle:** Blowing angles (the angle of the jet-pipes from the horizontal) of from 10° to 18° were used. This angle did not appear to be critical, but a median angle of about 14° seemed to work very well.

**Recoveries:** On such small-scale operation, it is difficult to obtain reliable figures on yields or recovery. Spills and metal lost in slagging operations are apt to be a larger portion of the total than in large-scale operation. Even the samples taken constitute a significant percentage of the total. By weighing all of the scrap and the samples taken, in addition to the bulk metal, the figures listed in table III were obtained. These indicate that at least 90 pct of the iron in the pig iron can be recovered in good operation. Over-blowing appears to be one of the most serious causes of low recoveries.

The principal loss of iron is by oxidation and most of the iron oxide formed enters the slag. If the slag weights were known, the amount of iron lost this way could be calculated from the slag analysis. It is



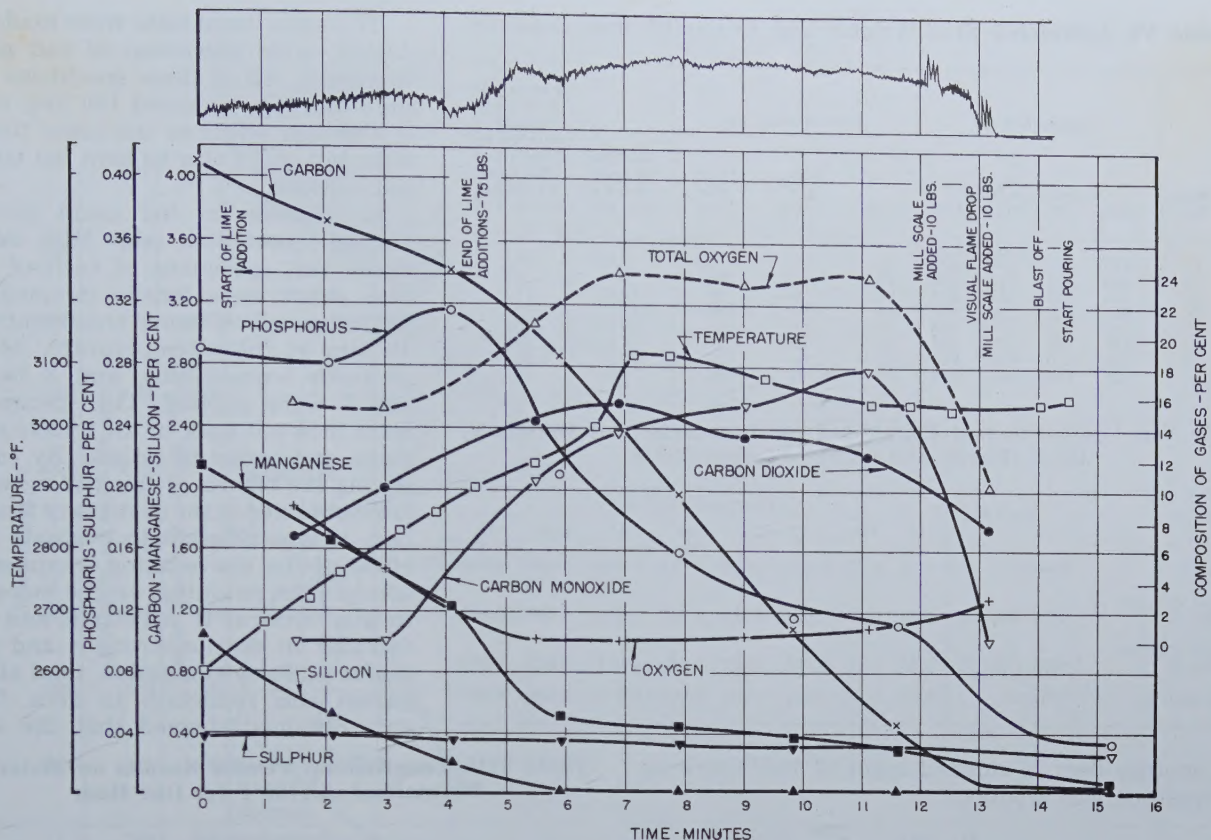


Fig. 7—Heat 25.

Tuyere velocity increased to 430 fps without change in volume. Very little effect on progress of blow.

even more difficult to obtain accurate slag weights than to get good metal weights. If it is assumed, however, that the only source of CaO is the burned-lime addition and that all of this enters the slag, then the slag weight may be estimated from the CaO content. The same thing can be done for silica assuming that the only sources are the burned lime and the silicon of the pig iron. MnO also has but one source, the manganese of the pig iron.

In table VI, slag weights of eight single slag heats have been estimated on the basis of analysis and available CaO, SiO<sub>2</sub>, and MnO, respectively. It will be noted that there is considerable discrepancy among the weights estimated from these three constituents in some heats, but there are also some good agreements.

From the average of the three estimated weights, the total iron content was calculated. After subtracting the iron added as mill scale, figures are obtained for the iron oxidized from the pig iron, both in pounds and as per cent. These indicate that it is possible to operate without losing more than 7 or 8 pct of the iron content of the pig iron to the slag. By comparing these figures with those of the last column, which represents iron loss calculated from the weights of pig iron and finished metal, it appears that mechanical losses played an important part in the recoveries.

**Refractory Life:** The experimental operation was not conducive to good estimates of the probable refractory life or cost. Only one heat was made in one day, and often there were several days or even a week between heats. A small gas flame kept the refractory lining from getting cold, but frequent cooling and heating cycles are especially hard on basic refractories. Some loss by spalling occurred in practically every heat, but it was mostly from the upper parts of the lining, well above the slag line.

Chemical erosion in the hearth area was very slight and there was no evidence at all of cutting at the slag level. When the original jets (of 1 in. steel pipe) were replaced after 24 heats, it was found that they had been shortened by 1½ in. and were still flush with the refractory surface.

**Fumes:** Compared to the Bessemer or Tropenas processes, the fumes given off during the blowing period in the new process were relatively small in volume. None was visible in the atmosphere of the laboratory building where the tests were conducted, for it remained clear at all times. Observations of the stack showed that some brown fumes were emitted for 2 to 3 min at the start of blowing (probably manganese oxide), but only thin white fumes were produced for the remainder of the heat.

**Large-scale Tests:** Following the experiments on the 1000-lb hearth, a series of large-scale heats were made at the South Works' Plant of the Carnegie-Illinois Steel Corporation.<sup>4</sup> This work was done in a hearth designed and built by the Jones and Laughlin Steel Corporation where it had been tested with an acid lining. A basic lining was installed at South Works and fifteen experimental heats were made with charges of 25 and 30 tons of hot metal, using fixed jets to surface-blow the metal.

Many operating problems still remain to be worked out before smooth practice can be established, but these commercial-scale tests completely corroborated the results of the laboratory tests as far as the chemistry of the process is concerned. For example, the iron and finished metal compositions on one heat were as follows:

	C	Mn	P	S	Si	N
Hot-metal analysis	4.39	2.08	0.228	0.031	0.96	0.003
Finished-metal analysis	0.015	0.037	0.008	0.019	0.003	0.002



Table VI. Calculated Slag Weights and Estimated Iron Contents

Heat No.	Estimated, Lb				Iron in Slag, Lb			Portion of Iron Added as Pig, Pct	Portion of Iron Lost as Calculated by Weights of Pig and Finished Metal, Pct
	CaO	SiO <sub>2</sub>	MnO	Avg.	Total	from Mill Scale	from Pig Iron		
11	218	230	268	239	80	7	73	7.8	19
13	184	238	244	222	72	7	65	7.0	9
23	284	280	311	292	108	14	94	10.0	11
24	338	335	320	331	120	14	106	11.5	15
25	275	238	274	262	77	14	63	6.9	18
27	347	202	277	242	63	11	52	6.0	10
28	508	310	430	416	162	14	148	16.0	25
30	326	258	291	292	86	19	67	7.4	12

Table VII. Composition of Plate Used for Comparison of Bessemer, Open Hearth, and Special Process Steels

Type	Heat No.	Composition, Pct							
		C	Mn	P	S	Si	Al	Al <sub>2</sub> O <sub>3</sub>	N
Bessemer	Y-4836	0.15	0.45	0.087	0.038	0.017	0.006	0.005	0.008
Special Process No. 1	4017, Heat 23	0.18	0.68	0.027	0.019	0.102	0.005	0.020	0.002
Special Process No. 2	4605, Heat 32	0.26	0.52	0.022	0.024	0.086	0.007	0.013	0.003
Open Hearth	72L083	0.26	0.47	0.014	0.034	0.075	0.006	0.009	0.003

On another heat, a rimmed ingot of the following composition was produced:

	C	Mn	P	S	Si	N
Hot-metal analysis	4.46	1.58	0.214	0.039	0.70	0.002
Ladle analysis	0.05	0.38	0.020	0.013	0.005	0.003

Three of the best 30-ton heats, those which had the shortest blowing times, used very close to 18,000 cf of air per ton of hot metal charged, which is in excellent agreement with the small-scale tests.

#### Mechanical Properties of Finished Product and Comparison with Those of Commercial Bessemer and Open Hearth Steel

During the experimental program, some of the heats were raised in carbon, manganese, and silicon by the addition of carbon and of ferroalloys and deoxidized by aluminum to produce ingots of semi-killed plate and structural grade. The one-half-inch test pieces which were processed from two of these ingots are designated in the following text as Special Process No. 1 and Special Process No. 2. The mechanical properties of these test pieces were compared with those of specimens from plates processed from a representative Bessemer blow and a representative open hearth heat. In selecting the material, the composition, particularly with respect to carbon and manganese, was specified to permit a comparison at approximately the same strength level after normalizing. Because the subject steels from the three processes were reduced to ½-in.-thick plates by methods which were not identical, it was not feasible to control the finishing temperatures, in the range desired. Hence, all of the materials were normalized for 1 hr at 1700°F before testing.

The specimens used for making tensile tests had reduced sections, ½ by ½ by 2¼ in. long, and gauge lengths of 2 in. This specimen size was adopted so that the existing extensometer equipment could be utilized; consequently, the per cent elongation is reported in a 2-in.-gauge length.

The chemical compositions of test specimens in per cent as obtained by check analyses appear in table VII.

The usual bend tests were made on 1½-in.-wide specimens of full plate thickness. All of these specimens not only bent the required 180 deg. over a diameter equal to the plate thickness, but could also be bent flat without rupture.

In addition to the usual tensile testing reported in table VIII, additional test specimens of each of the four steels were tensile strained 10 pct after which aging treatments of 10 min at room temperature, 24 hr at room temperature, and 3 hr at 400°F were applied. The specimens were then put back in the tensile machine and tested to failure. By comparing the results of this testing with those obtained in the customary tensile test, it was possible to evaluate the effect of the above aging treatments, after 10 pct straining, on the increase in true stress at 10 pct strain, and the increase in tensile strength and the loss in uniform elongation, total elongation and reduction in area. The data obtained showed that the two

Table VIII. Longitudinal Tensile Results on Materials Normalized at 1700°F for One Hour

Tensile Properties	Bessemer	Special Process No. 1	Special Process No. 2	Open Hearth
Upper yield, psi	41,850	35,450	36,550	36,300
Tensile strength, psi	61,050	60,450	66,850	64,550
Elongation in 2 in., pct	37.5	35.7	35.2	37.5
Reduction in area, pct	64.7	56.8	57.7	61.6

special process heats performed similarly to the open hearth heat and both steels performed differently from the Bessemer steel. The same differences were brought out by hardness tests conducted before and after both straining and strain aging.

Another more commonly used test for determining strain aging was tried on the steels tested. This is the familiar concrete-bar test in which test specimens are bent 90°, aged at 550° for 1 hr and unbent. In this test, the Bessemer steel fractured with practically no deformation on unbending, whereas, specimens of the two special process steels and the open hearth steel unbent to a flat position without developing cracks.

Transition temperatures were determined on all of the materials in the normalized condition using Charpy keyhole notch-bar impact specimens. It was found that the special process steels were comparable to the open hearth steel and had lower transition temperatures than the Bessemer steel.

Charpy notch-bar impact tests were also conducted on these materials in both the strained and strain-aged conditions. The results indicate that the effects of both straining and strain aging are greatest in the Bessemer steel, and that the special process steels are much more like the open hearth steel in this respect.

One method sometimes used to evaluate strain aging is to compare the tensile strength at room temperature with that at approximately 400°F. Materials showing a gain in tensile strengths at



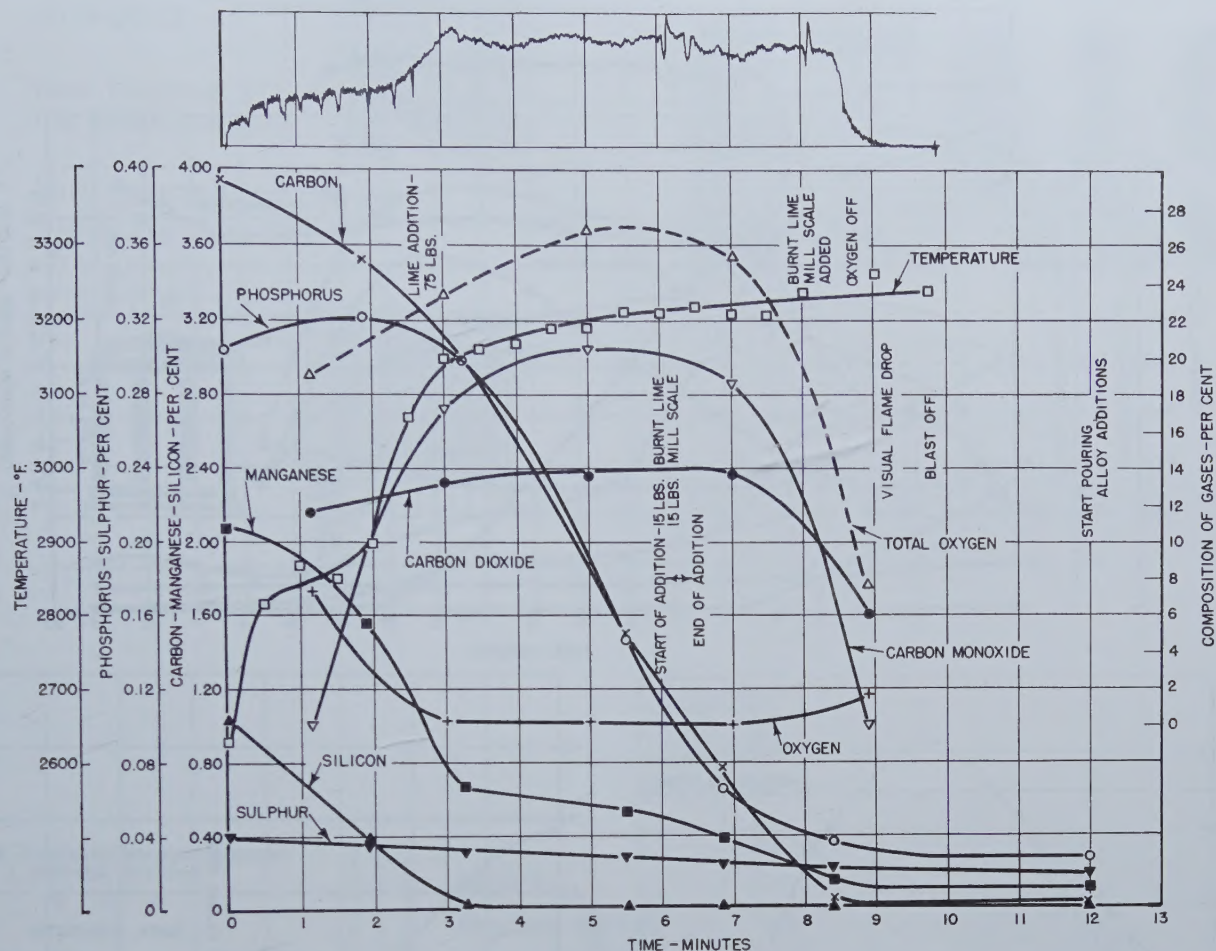


Fig. 8—Heat 27.

Blast air enriched to 25.6 pct  $O_2$ . Blowing time decreased 25 pct and temperature raised markedly.

400°F are said to be susceptible to strain aging, and the degree of strain aging is judged by this increase. The strain-aging propensity of the two special process steels in this test is comparable to that of the open hearth steel.

Heats made later at South Works by the special process, cast as ingots and rolled into plates were subjected to a number of the above tests (ordinary and extraordinary) to determine whether the properties differed from those of open hearth steels. It should be realized that the compositions were such as to give "as-rolled" tensile strength ranging from 45,000 to 49,000 psi, so that a direct comparison with the properties of the Battelle heats could not be made. However, the customary tests indicated that the tensile and bending properties of the South Works special process steels were equivalent to normal open hearth steel properties for a similar composition. In addition, the reverse bend tests indicated clearly that the material performed as well as open hearth steel. The Charpy transition temperatures were favorable and the amount of strain aging exhibited in the Charpy tests was approximately the same as that previously exhibited by the Battelle heats. It was, therefore, concluded that this special process material behaves in a like manner to open hearth steel of similar composition.

### Summary and Conclusions

Laboratory tests on 1000-lb charges of hot metal showed that surface blowing in a basic-lined hearth to produce steel from basic pig iron is chemically sound and metallurgically feasible.

No special compositions of hot metal are required, standard basic pig iron is entirely suitable.

Carbon flame-drop was abrupt and was an accurate indication that the carbon content was close to 0.05 pct.

The time of heat was a function of the rate of air input or of oxygen input when oxygen-enriched air was used. Average process time was 12 min. The total air requirement was at the rate of 18,000 of per ton of hot metal and blast pressures of less than 3 psi were used.

A substantial part of the temperature gain was found to be from the oxidation of about half the carbon to  $CO_2$ .

Carbon can be oxidized to below 0.03 pct, manganese below 0.1 pct, and silicon below 0.01 pct.

Nitrogen can be maintained at about 0.003 pct.

Up to 97 pct of the phosphorus and up to 50 pct of the sulphur can be eliminated. Phosphorus and sulphur were removed by the time the flame dropped, and no afterblowing was necessary or desirable.

Burned-lime requirements were from 120 to 180 lb per ton.

The fumes produced were definitely less dense than in the acid Bessemer process, and slopping was controlled to the point where it was no problem.

Commercial-scale tests, using as much as 30 tons of hot metal, corroborated the findings of the laboratory experiments.

Mechanical properties of the steel produced are such as to place them in the same limits as open hearth steel of a similar composition.



